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APPLICATION OF MINERAL BENEFICIATION TECHNOLOGY TO COAL PREPARATION

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ABSTRACT

Advances in mineral processing technology and equipment are described and their applicability to coal preparation is outlined. Some advances in coal preparation are also discussed since these may have direct relevence to the cleaning of Eastern Canadian coals. Techniques applicable not only to conventional coal preparation but those suitable for intensive cleaning, sulphur reduction and coal-liquid mixtures are also included. Recommendations are classified under laboratory research and tests, pilot scale tests and full scale plant tests. The priorities for future work are also suggested.

1 INTRODUCTION

In the past, coal preparation was mainly concerned with the rejection of shale from relatively coarse run-of-mine coals and with screening to consumers' requirements. Coal fines were present in lower proportions and were regarded as an unimportant part of the total production. With extensive use of mechanised mining methods the fines content of the mined coal has substantially increased and thus fines have become an important part of output. Current coal treatment methods are required to deal with this increased fines content of run-of-mine coal. An additional restraint arises from environmental protection considerations as it is becoming unacceptable to burn coals of high sulphur content. It is not too difficult to adequately reduce ash content but sulphur reduction is more complicated due to the fine distribution of sulphur in coal. For improved sulphur removal pyrite liberation by grinding is necessary. In this intensive coal treatment, new technologies must be employed, which are capable of operating at finer sizes, with adequate selectivities of separation and with economic viability. In intensive coal treatment, solid/liquid separation of fine products also involves additional costs and treatment complexities.

Mineral processing technology has solved similar problems and, therefore, examination of the various technological innovations used in the treatment of minerals and evaluation of their applicability to problems in coal treatment should be of considerable value. Mineral processing activities are very diverse, varying from concentration of high valued products such as precious metals and diamonds to medium priced metals like tin, nickel and copper and to low priced products including fluorspar, salt and gypsum. The feed grades of the ores treated may vary from a few parts per million to over 50%. Some ores are concentrated at coarse sizes while others must be finely ground before concentration of minerals can begin. Although fine grinding is costly, a relatively cheap iron ore concentrate can be produced competitively when the ore is ground to 75% finer than 32 µm. Mineral processing practices have shown that even a sophisticated treatment can be carried out economically when operating at high feed rates in a properly designed plant.

In this report an attempt has been made to find possible applications of innovative mineral processing techniques to coal treatment by systematically examining the unit operations of mineral processing. The references surveyed were published in journals, text books, papers presented at congresses and symposia, various reports and manufacturers' specifications. Articles collected by the author for use in compiling the Mineral Processing section of Mining Annual Review for the last nine years formed the nucleus of the technical information presented here.[1-9] Time and space have limited detailed discussion of individual topics but the references given provide more detailed information. While a broad coverage of topics has been sought, emphasis has been placed on the areas showing greater promise of adaptation to coal treatment. No claim is made that the coverage of this report is complete. Although based on references collected from the relevant fields the conclusions are the personal opinions of the author and, therefore, may be biased.

2 DISCUSSION

The most important coal reserve in Eastern Canada is the Sydney Coal Field, containing an estimated 3 billion tonnes of coal. The quality of the coal varies greatly, even within one seam. The ash and sulphur contents tend to increase towards the eastern and western extremeties of the field and to be lowest in the central portion.[10] The ash content varies between 2% - 18% and the sulphur content is between 1% - 6%. The Victoria Junction Coal Preparation Plant started operation in 1976, treating No. 26 coal and Lingan coal.[11] About 72% of yield is metallurgical coal with 2.7% - 2.9% ash and 1.2% - 1.3% S and 12% of yield is thermal coal with 10% ash and 2.5% S. Although the sulphur content of the coking coal is not low by current industrial standards, the coal has a low ash content and about 40% of the ash content is iron oxide which is advantageous in steel making, giving an effective ash content of only 1.6% - 1.7%. Other coalfields in Nova Scotia include Port Hood with reserves of 35 Mt, Pictou with 50 Mt and Springhill with 50 Mt.

New Brunswick has a probable reserve of 50 Mt of highly volatile bituminous coal.[12] Its ash content varies between 14% - 20% and its sulphur content between 6% - 8%, which includes approximately 2% of organic sulphur and minor amounts of sulphate sulphur. Its main use is as thermal coal.[13] Due to the fine distribution of pyrite in Minto coals, only moderate sulphur reduction may be achieved by conventional washing as indicated by float and sink tests. When considering the applicability of mineral processing techniques to the treatment of Eastern Canadian coals the main characteristics of these coals must be continuously borne in mind as they determine the suitability and applicability of techniques.

2.1 Liberation - Crushing and Grinding

Liberation

Particles of an ore may consist of a single mineral in which case they are called "free particles" or two or more minerals in which case they are known as "locked particles". A prerequisite of good separation is that the ore is mainly composed of free particles. The proportions of free particles may be increased by crushing and grinding. In the 1930's Gaudin laid down the fundamental principles of liberation. He defined degree of liberation of a certain mineral as the percentage of that mineral occurring as free particles in relation to the total of that mineral.[14] Gaudin recognized that the main variables affecting liberation of a mineral are: the grain size of the mineral in the uncrushed rock, the particle size after crushing and the relative volumetric abundance or volumetric fraction of mineral in the rock. Through analysis of a simplified three-dimensional model he concluded:

1. The less abundant mineral is not freed at all unless the particles are finer than grain size.

2. Particles must be much finer than grain size if the less abundant mineral is to be freed substantially.

3. When the relative abundance is great, the more abundant mineral may be appreciably freed even when the particles are coarser than grain size.

4. The more abundant mineral is always freer than the less abundant mineral.

The above conclusions may be interpreted for the liberation of raw coal as follows. Since coal is more abundant than shale the degree of liberation of coal is higher than that of shale. As the grain size is large, substantial liberation occurs at relatively coarse sizes. The coarse shale, which is mixed in during mining, is also reasonably liberated. But the finely divided mineral matter distributed in the coal is practically impossible to liberate due to its fine grain size and low relative abundance.

Sulphur may be present as organic sulphur, pyritic sulphur and often as insignificant amounts of sulphate sulphur. Since organic sulphur is fairly evenly distributed throughout the coal, it is not liberated by crushing and grinding and thus cannot be separated by physical methods. A good proportion of pyritic sulphur is usually finely distributed and this fraction is difficult to liberate. More coarsely distributed pyrite may be liberated to some degree and rejected by separation methods. Even fine grinding will not adequately liberate finely divided pyrite present in coal in very low volumetric proportions. (Note: 1% pyritic sulphur equals 1.47% pyrite by weight and 0.42% pyrite by volume.) But, since coal is liberated to a much greater degree, pyritic sulphur reduction may be achieved at the price of losing some coal as locked particles with pyrite.

Liberation by size reduction is the standard method. In some cases mineral characteristics permit additional liberation by detachment on mineral boundaries. Liberation by detachment relies on intergranular weaknesses or differences in the physical properties of adjacent minerals. These are characteristics of the ore and cannot be significantly modified. Parting of coal from shale or coal from platy pyrite will increase liberation. The physical properties of minerals will also influence the make-up of locked particles. Locked particles are often referred to as middlings and are the main cuase of separation difficulties. Hard mineral grains (e.g. pyrite) distributed in a softer mineral (e.g. coal) often result in locked particles in which the harder mineral is enclosed in a shell of the softer mineral due to the preferential fracturing of the latter. In density separation these locked particles behave according to their mean densities while in magnetic separation they are separated by their overall magnetic properties. In flotation the mineral shell material determines to which product these particles report, as it does in electrostatic separation. Therefore, knowledge of locked particle behaviour is important for prediction of separation results.

Gaudin outlined quantitative methods for the evaluation of degree of liberation but since the mineralogical work involved in the estimation of liberation characteristics is very laborious, not much progress was made until computer assisted microscopic techniques became available in the 1970s.[15] The use of image analysis techniques with the aid of scanning electron microprobe, which is able to identify individual minerals, provides rapid data acquisition. Two types of automatic image analysis equipment are available, one, the area scan analyzer, examines two dimensional sections of the structure while the other, called linear scan analyzer analyses images on the basis of a linear scan-line.[16,17] The data obtained from image analysis must be interpreted in terms of a three dimensional structure. The stereological relationship of locked particles to the two dimensional image always relies on simplifying assumptions for the complex structure of the actual locked particles.[17,18,19] The interpretation of linear scan data is said to be simpler than for area scan. Random structure and equally brittle minerals are usually assumed. The interpretation becomes complicated when preferential fracturing on boundaries is involved.

At the Commonwealth Scientific and Industrial Research Organization (CSIRO), Division of Mineral Chemistry, Port Melbourne, Australia, QEM*SEM (Quantitative Evaluation of Minerals by Scanning Electron Microscopy) image analysis has been used in mineral liberation studies.[20] This method uses linear scanning in which the number of transitions from one mineral to another is also determined. From the data obtained the "normalized association probability", which defines the degree of randomness of mineral associations and the "cumulative liberation yield" are calculated. The latter expresses the yield of a given mineral carried in those particles for which the volume fraction of the mineral is greater than a given value.

Determination of liberation characteristics of coal feeds by means of quantitative mineralogical and petrological analyses is of great importance in the optimisation of coal treatment. This area deserves greater emphasis as it affects all aspects of coal treatment, especially when finer treatment sizes are involved.

Crushing

The primary aim of coal crushing is to increase liberation of coal without the production of excessive amounts of fines. Although the type of crusher partly determines the proportion of fines produced, operating conditions are often more important. It is well known that choke feeding of a crusher creates more fines than arrested feeding. A particular type of crusher is only efficient on a certain size range of feed.[21] To reduce fines production, the fines should be promptly removed from the feed and the crusher should be operated in closed circuit with a screen.

There is a wide variety of crushers from which to choose. Gyratory and cone crushers have not been used much in coal preparation as these machines were designed for hard rock.[22] Jaw crushers are suitable for coal crushing as they produce cubical products with a minimum of fines. The most popular coal crushers are toothed-roll crushers. They are made in double roll form where the two counter-rotating toothed-rolls crush the coal inbetween the rolls.[23] One roll is normally spring loaded as a counter measure against severe shock or breakage due to tramp material. Alternatively, toothed-roll crushers are available as single toothed-roll models which act on the feed material between a rotating roll and a curved plate, (Figure 1). Here the curved plate has some sort of mechanism for the release of tramp material. The wedge action of the teeth on these crushers minimizes fines production. Single roll crushers are also useful for breaking up frozen coal, (Figure 2). These crushers have extra wide feed openings to permit direct feeding from hoppers. They reduce frozen coal to below 150 mm - 200 mm. The breaker plate may be backed off during warmer weather to allow direct flow through the unit.

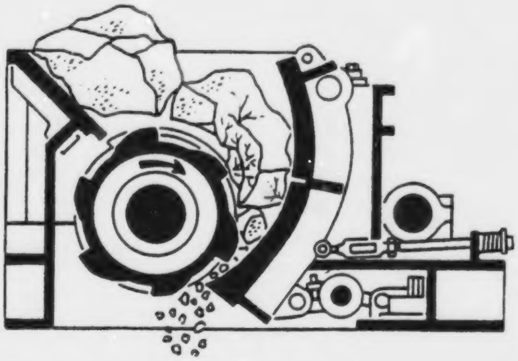


FIGURE 1: Single toothed-roll crusher (Pennsylvania Crusher Corp.)

Hammer mills break coal by the impact of rotating hammers throwing the coal against breaker bars, followed by hammers dragging the coal against screen bars, which result in sheer and attrition, (Figure 3). Material which does not pass through the screen bars on the first occasion swirls around

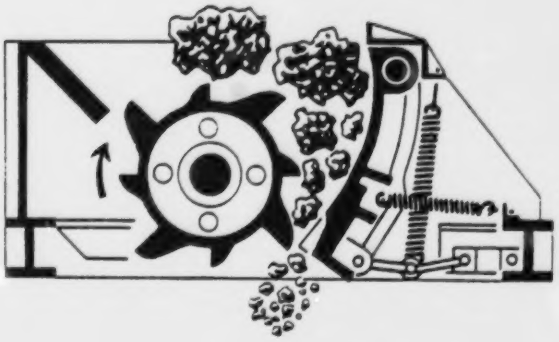


FIGURE 2: Frozen coal cracker (Pennsylvania Crusher Corp.)

until it is below grate opening size. This type of hammer mill tends to produce large amounts of fines. A variation of the hammer mill is the ring type crusher or granulator which nips particles between screen bars and circular hammer heads, (Figure 4). They produce fewer fines.

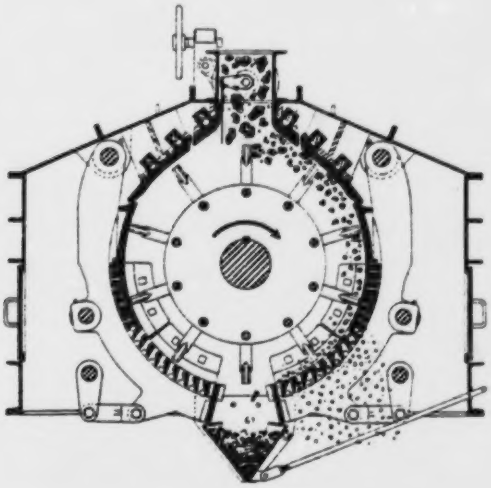


FIGURE 3: Reversible hammer mill for coal (Pennsylvania Crusher Corp.)

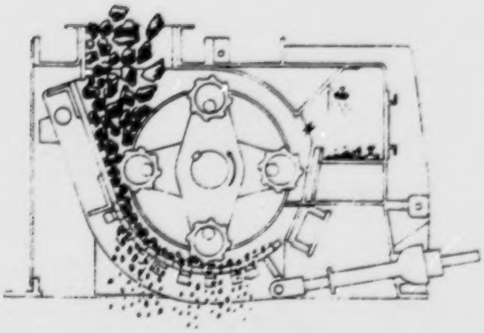


FIGURE 4: Granulator (Pennsylvania Crusher Corp.)

Impactors reduce by striking coal with hammers or impeller bars against breaker blocks, (Figure 5). These machines have no precise top size limit for the product but they produce less fines than hammer mills. Some impactors are reversible to even out wear on hammer faces and breaker blocks. A special impact crusher is equipped with revolving grates, (Figure 6). The grates provide a breaking surface and also act as a screen, ensuring that any coal finer than the grate aperture is promptly removed from the crusher.[24] Thus, this machine gives less fines and a well defined product top size.

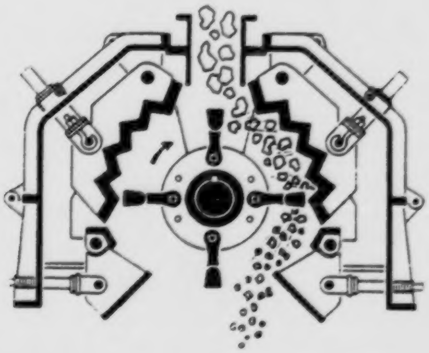


FIGURE 5: Impactor (Pennsylvania Crusher Corp., Coalpactor)

The Roller Screen Crusher is designed to crush coal in two stages and to screen over roller screens.[25,26] Both crushing rolls are counter-weighted so that they leave hard

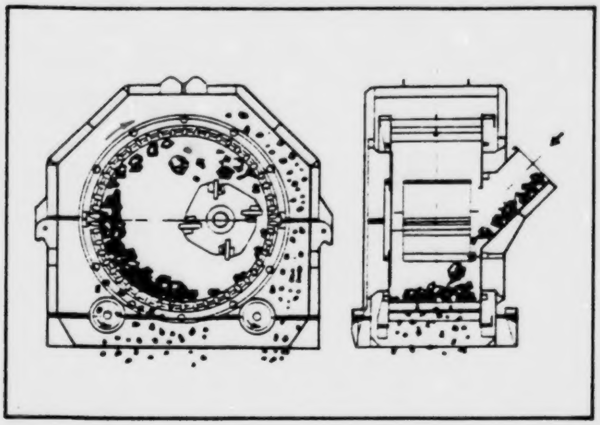


FIGURE 6: Revolving grate impact crusher (Indutec GmbH & Co.)

material uncrushed and it is rejected from the roller screen. Thus, these crushers separate hard shale from raw coal, (Figure 7). The product size may be adjusted to between 20 mm - 60 mm. The largest unit is 2 m wide and has a capacity of 4000 t/h.

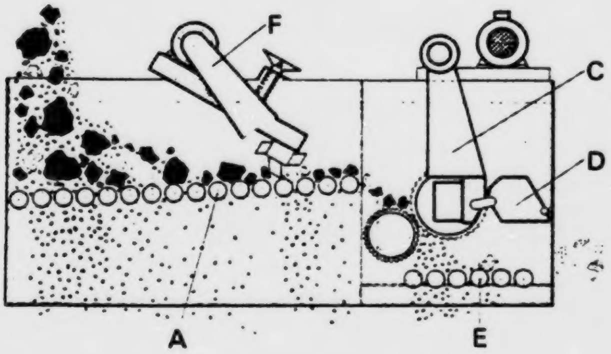


FIGURE 7: Schematic diagram of Roller Screen Crusher (Kone Corporation, Finland)

Rotary breakers, which are variations of the original Bradford breaker are offered by several manufacturers.[21]

In these machines, lifters in the trommel raise the material and allow it to fall and thus gravity impact causes particle fracture. The crushed coal passes directly through the trommel screen while uncrushable material is scooped out by a refuse plough at the end of the trommel. Two main types are available: trunnion mounted and roller mounted rotary breakers. The top size of feed to the trunnion mounted version is limited.

Feeder breakers are often used underground in coal mines to reduce large lumps to 250 mm - 300 mm maximum size for transportation by conveyors.[27] Feeder breakers are also being used in surface mining as the tendency is towards conveyor belt transport instead of more expensive truck haulage. They ensure a steady loading of conveyors without large lumps. Feeder breakers are available with capacities between 200t/h - 4000t/h.

Water flushing of coal during grinding is reported to reduce both fines production and power consumption and to increase throughput.[28] The technique, referred to as hydrocrushing, was tested on a jaw crusher with a 150 mm feed opening. The water flush removed fines from the crusher as generated, thus considerably increasing throughput and reducing power consumption by one third. Hydrocrushing is not yet a practical possibility because of the large quantities of water needed.[22]

The selection of the most suitable crusher for a certain duty is not simple as there are many types available with varying characteristics. The most important factors which should be satisfied by the crusher are: throughput capacity, maximum feed size, required product size, product size distribution, material characteristics such as moisture content, stickiness and abrasiveness.[29] Sticky and clayey feed may clog the crusher and prevent efficient operation. The abrasiveness of the feed is often due to its silica content, which is the main cause of machine wear.

A new book, published by the Mineral Processing Division of the Society of Mining Engineers of AIME, deals with the Design and Installation of Comminution Circuits.[30] The 61 chapters, written by acknowledged experts and totalling over 1000 pages, comprehensively cover all aspects of crushing and grinding technology. It is not possible to summarize the contents here, but for those engaged in various areas of comminution and classification it is an essential reference book. Another useful volume on Mineral Crushing and Grinding Circuits is by Lynch and gives good coverage of their simulation, optimisation, design and control.[31]

Grinding

While the term crushing refers to coarser comminution processes, grinding is reserved for the finer end of size reduction. Grinding has not been considered as an important part of coal preparation and has only been used in special circumstances such as the preparation of feed for pulverized fuel combustion. However, it is expected that the importance of coal grinding will increase as coal-liquid fuels gain acceptance in industry and as pyritic sulphur removal will necessitate intensive coal cleaning. Since for either purpose, the coal product should not contain large proportions of ultra fines, suitable and cost effective grinding methods will have to be evolved. In mineral processing grinding is usually the highest cost operation and therefore considerable theoretical and industrial research has been conducted into the reduction of grinding costs and in improving product characteristics. The knowledge gained from this research should be of benefit when applied to coal.

Ring and ball mills have been used efficiently for the dry grinding of coal for some time. These mills grind the coal by hollow balls running between an upper stationary grinding ring and a lower rotating grinding ring. An adjustable pneumatic classifier is usually incorporated in the unit. A representative model of ring and ball mills is the Babcock E mill capable of grinding up to 150 t/h coal to 70% below 150 μm.[32] The largest model designed operates with ten 914 mm balls. Advantages of this model include low noise and vibration levels, low power consumption and easy replacement of worn balls and grinding rings. Roller mills operate on very similar principles to ring and ball mills and have been used for pulverisation of coal for power station use.[33]

Cage mills or disintegrators are internally fed impactors. In these mills, two, four or six concentric cages rotate with adjacent cages rotating in opposing directions.[34] The material to be dry ground is fed into the innermost cage and the grinding action takes place by repeated impacts. Cage mills are available from 0.45m - 2.6 m diameter. Disintegrators are suitable for grinding coal. The largest feed size accepted by a large disintegrator is around 100 mm. Ideally, disintegrators are operated in closed circuit with either a screen or a pneumatic classifier to lessen fines production.

The Szegő mill is a vertical mill with a stationary grinding cylinder in which three helically grooved rollers rotate on flexible wire ropes,[35] (Figure 8). The rollers are pressed against the grinding cylinder by centrifugal forces and the mill may be operated wet or dry. The Szegő mill is said to compact, to have high capacity and modest power consumption. It is suitable for fine grinding.

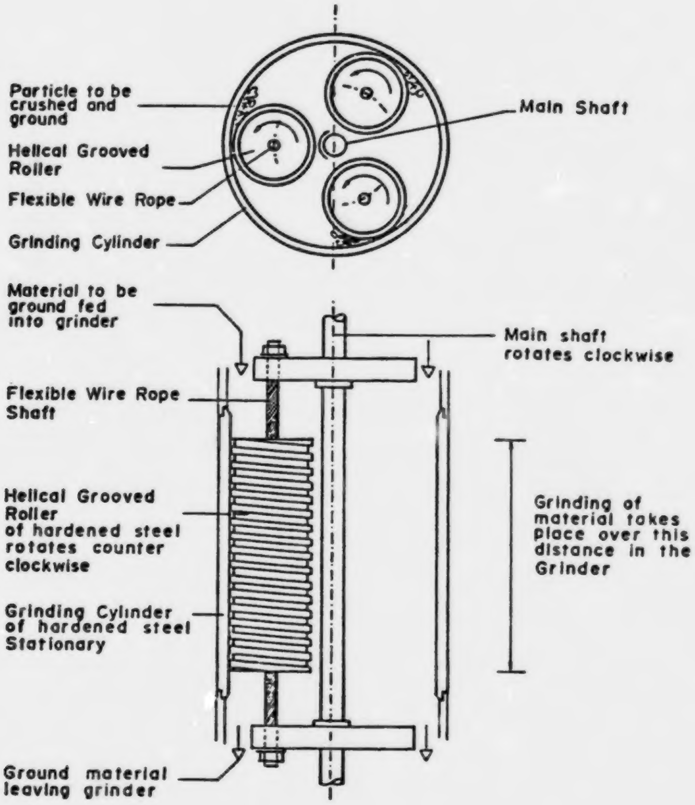

Particle to be crushed and ground
Main Shaft
Helical Grooved Roller
Flexible Wire Rope
Grinding Cylinder
Material to be ground fed into grinder
Main shaft rotates clockwise
Flexible Wire Rope Shaft
Helical Grooved Roller of hardened steel rotates counter clockwise
Grinding of material takes place over this distance in the Grinder
Grinding Cylinder of hardened steel Stationary
Ground material leaving grinder

FIGURE 8: Schematic diagram of the Szegö mill.

In mineral processing, rod, ball and autogenous mills are the most important types. While rod and ball mills have steel grinding media, in autogenous mills larger lumps of ore grind the smaller particles. Rod mills grind in line contact between rods and their action is similar to that of a smooth roll crusher. Rods grind larger particles preferentially, resulting in a narrower product particle size distribution with less fines. Balls, on the other hand, grind by point contact and show little selectivity towards larger particles. For this reason ball mills are usually operated in closed circuit with a classifier, which removes the fines from the mill product and returns the coarse fraction for further grinding. Although rod and ball mills may be operated dry they are most frequently used for wet grinding. The amount of fines generated in wet grinding depends on the speed at which the pulp flows from the feed end to the discharge end of the mill. For an overflow discharge mill the flow rate is slow and so more fines are produced.[36] In a diaphragm discharge mill the pulp flow is faster but the peripheral discharge types give the fastest pulp flow producing least fines.[37]

Autogenous mills, commonly used in mineral processing are manufactured up to 11 m in diameter and 5.2 m long. The energy consumption of an autogenous mill per tonne of feed ground is about 50% more than in ball or rod mill grinding. In spite of the greater energy cost, autogenous mills can compete with conventional mills as an autogenous mill can replace two crushing and one grinding stages and needs no grinding media. As they require specific breakage characteristics of the feed material and operate best with higher density mineral feeds, 2.5g/cm^3 and above, autogenous mills do not appear to be appropriate for use with coal. However, autogenous grinding could be utilized for the preferential grinding of coal constituents. The harder lithotypes would appear as coarser particles in the ground products than the softer lithotypes, making their separation possible.

Vibrating mills are suitable for the very fine grinding of a wide range of materials. Feed sizes of 25 mm - 30 mm may be reduced to 5 μm.[38] The PALLA 65 U vibrating mill manufactured by KHD Humboldt-Wedag has been used for grinding brown coal at 9 t/h for pulverised coal firing.[39] The drive mechanism is an unbalanced rotor running at 1500 rpm. In this case rods rather than balls were used as grinding medium.

The concept of centrifugal milling is not new as the process was described in a US patent of 1896. So far, centrifugal milling has not found full scale industrial application. In general a centrifugal mill consists of a cylindrical mill rotating around its axis while simultaneously gyrating around a parallel stationary axis. Approximately eleven years ago the South African Chamber of Mines built and tested laboratory scale centrifugal mills for ore grinding. There it was found that centrifugal mills had high throughputs but excessive liner and grinding media wear.[40] More recently, a joint effort by the South African Chamber of Mines and Lurgi of West Germany has been started to develop large scale centrifugal mills.[41]

In centrifugal milling centrifugal acceleration is utilized instead of acceleration due to gravity. Since centrifugal acceleration may be increased as desired the grinding forces will also be increased proportionately. The power drawn by a conventional mill is proportional to its length multiplied by its diameter to the power of 2.5 In centrifugal milling the power drawn is proportional to the product of its length, the diameter of gyration, the mill diameter cubed and cube of the speed of gyration. This represents a significant increase in power drawn and consequently in throughput. Power densities of 1000 to 1500 kW/m^3 of mill volume are said to be possible.

In the new Chamber of Mines / Lurgi design the mill does not rotate but only gyrates. This configuration permits a compact design, with simple charging and discharging arrangements.

This type of centrifugal mill may be considered as a vibrating mill with a very large amplitude, but the mill charge is not fluidized as in vibrating mills. The 1 m diameter by 1.2 m long prototype built draws 1400 kW and grinds material from 100 mm to 55% finer than 70 μm at a rate of 23t/h when operating at 230 gyrations/min.[42,43] The advantages of centrifugal milling arise from the lower weight, lower space requirements, ease of manufacture, lower capital cost and easy maintenance. In South Africa it is intended to use the mill underground. If the tests are successful, consideration of using centrifugal milling for coal would be worthwhile.

The Tower mill was originally developed in the 1950s in Japan, where it was used for grinding a variety of materials.[36,44,45] More recently Koppers Mineral Processing Systems Division has manufactured the mill under licence. A schematic diagram of the mill is shown in Figure 9. The mill consists of a vertical, cylindrical grinding chamber (1), a screw flight agitator (3), driven by a motor through a gearbox (2). The grinding chamber contains steel balls or pebbles and the feed which enters by a chute. Ground particles are carried upwards and overflow into a coarse classifier (4) from which the coarse product enters a sump (5) and is returned by a pump (6) for further grinding. The fine product of the coarse classifier enters a sump (7) and is pumped (8) to a hydrocyclone (9). The hydrocyclone overflow is the finished product while the underflow is returned to the mill via a sump (5). The manufacturers claim several advantages for the Tower mill over conventional ball mills. The Tower mill uses only half the energy when grinding in the 1 μm to 100 μm range. It has lower initial costs, easier installation and requires less space. It is quieter and produces less overgrinding. Harder steel balls may be used because there is no tumbling action. The

Tower mill is particularly suited for fine grinding such as occurs in regrinding circuits. A 200 kW Tower mill has been installed in the Macassa mill, Kirkland Lake, Ontario, to grind to 95% - 38 μm.

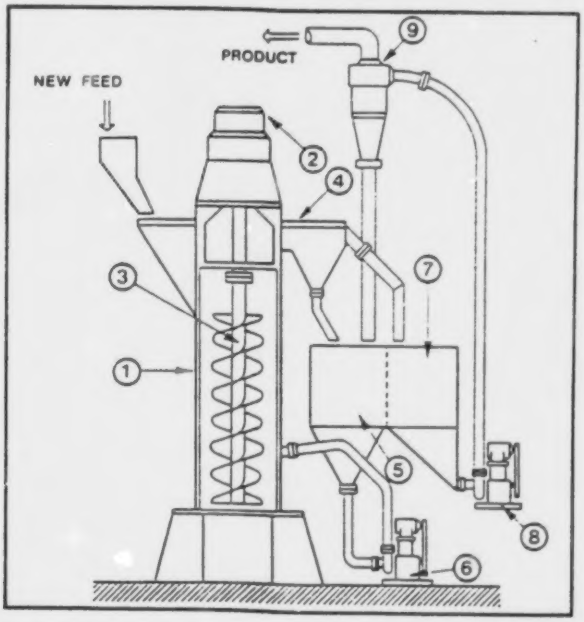


FIGURE 9: Schematic diagram of the Tower mill.

The Paddle mill, manufactured by Magstone Development Inc., Derbert, Nova Scotia, operates on similar principles to the Tower mill. It has a vertical rotating shaft equipped with paddles in place of the Tower mill's screw flight agitator. The paddles rotate at 300-400 rpm. The feed enters near the base of the grinding chamber and leaves at the top. The grinding medium may be replenished during operation when necessary as indicated by a drop in motor current. The paddles need replacement after about 4000 h. Current production models are offered with power inputs up to 75 kW. This mill is being tested by the manufacturers for suitability for fine grinding coal.

Most grinding mills are lined with replaceable steel liners but the use of rubber mill liners has increased since the 1960s. The mechanical and wear properties of rubber liners differ greatly from those of steel.[46] With steel liners the wear is caused by a combination of corrosion, impact and abrasion. Rubber liners do not corrode under normal mill conditions but may be attacked by oils when oil is present in excess of 0.5 kg/t of feed solids. At low impact speeds, the deformation of rubber is entirely elastic but above 9 m/s surface damage may occur. While steel liner wear increases rapidly with mineral hardness, rubber liner wear is almost unaffected by mineral hardness. Low angle attack and sliding of mill charge should be avoided to prevent excessive wear of rubber liners. Correctly designed rubber liners should give at least the same mill capacity and grinding efficiency as steel liners, should improve liner economy and reduce mill noise.

Orebed magnetic lining introduced by Trelleborg AB (in Canada and U.S.A., marketed by Orebed Inc.), is another method for wear reduction of liners: in fact according to the manufacturers the liner never wears out.[47,48] The principle of Orebed magnetic lining is shown in Figure 10.

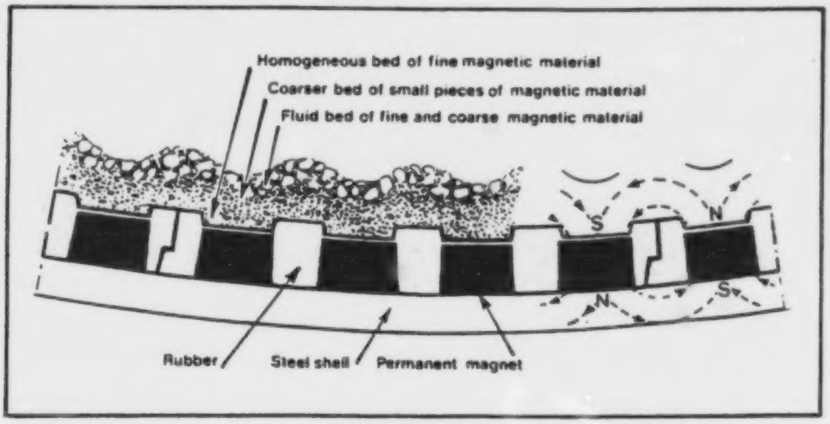

Homogeneous bed of fine magnetic material
Coarser bed of small pieces of magnetic material
Fluid bed of fine and coarse magnetic material
S
N
N
S
Rubber
Steel shell
Permanent magnet

FIGURE 10: Orebed magnetic lining

The flexible mill liner has permanent ceramic magnets vulcanized to the rubber. The liner does not require bolting since the magnets hold it in place.

Any magnetic mineral in the ore feed or derived from the grinding medium becomes attached to the liner by magnetic attraction, forming a wear protecting layer which is continuously renewed. Orebed magnetic liners could be used with advantage in ball mills used for grinding magnetite for dense medium separation of coal. Before the ground dense medium is sent to the sump it should pass through a demagnetising coil.

In evaluation of the grindability of an ore it is common practice to determine the Bond work index of the ore, which is the energy in kW/h required to reduce one short ton of ore from theoretically infinite size to 80% passing 100 μm. With the work index known it is possible to evaluate the energy required to grind from a given 80% passing feed size to a required 80% passing product size. The prediction is good, but the laboratory work involved in the determination of the work index is substantial. For coal, the Hardgrove grindability index is used, whose determination is relatively simple. McIntyre and Plitt correlated the Bond work index with the Hardgrove grindability index, thereby making it possible to predict the Bond work index from the Hardgrove mill tests.[49] They found good correlation between the two measures. This approach simplifies the work of mineral processors by using techniques originally developed for coal preparation. Perhaps, when power consumptions of crushers and grinding mills will be predicted for coal, coal preparation will also benefit.

The selection, sizing, simulation and control of grinding circuits are fully covered in text books.[30,31,50] Especially up-to-date and full coverage is given in reference 30, "Design and Installation of Comminution Circuits".

Numerous attempts have been made to reduce grinding energy requirements by the use of chemical additives. The significance of the problem becomes clear when it is realised that about 50% of the annual 100 billion kWh energy consumed in U.S. mineral processing plants is used in comminution and only about 0.6% of the grinding energy is expended in creating new surfaces.[51] Numerous chemicals have been tested in the grinding of many ores with varying degrees of success. Workers generally agree that grinding energy reduction is connected with improving the rheological properties of pulp in the mill.[52→57] Coagulated and flocculated pulps tend to consume more energy than dispersed pulps. There appears to be an optimum pulp dispersion or viscosity which gives the most efficient grinding. The XFS-4272 grinding aid, marketed by Dow Chemical Company, is the most commonly used. This is a low molecular weight, water-soluble anionic polymer with selective dispersant properties. In a two part paper, Klimpel outlines the influence of slurry rheology in mineral and coal grinding circuits and gives guidelines for industrial evaluation and use of grinding aids.[58,59] Grinding aids could give significant savings in the wet grinding of coal for coal-liquid fuels and for the Carbogel process.

2.2 Screening

Screening is a commonly used size separation method. The fundamental principles of screening have been given by Gaudin, who considered the probability of passage of particles through screen apertures.[14] From Gaudin's work, it becomes clear that particle sizes close to screen aperture size have a very low probability of passage through the screen. Practical experience confirms this. Particles somewhat coarser than screen aperture size tend to plug or blind apertures and thus reduce the useful area of the screen. Blinding of screen apertures is a major drawback and many methods have been devised to lessen the problem. Although the basic design of most industrial screens has changed little for several decades, many detailed improvements have been incorporated in new machines and some screens employing interesting new concepts have been introduced. Improvements in vibrating screens include: new vibrating mechanisms, variation in the type of vibration, use of fabric reinforced or foam filled rubber cushions for supports instead of steel coil springs and improvement in design and fabrication which allow the manufacture of larger and wider units without risk of structural failure.[60]

Free-swinging, "two bearing" vibrating screens are the most popular. They are suitable for a wide range of applications in wet and dry screening and are made in single to six deck designs. These screens are usually inclined 10^o-30^o to the horizontal to aid the flow of material. Their circular vibration is generated by a motor driven shaft with a bolted on unbalanced weight. The direction of shaft rotation influences screening results. The more expensive "four bearing" types of vibrating screen give a forced motion with constant amplitude. Thus their deck vibration is unaffected by the amount of material on the deck. "Linear motion" screens employ two unbalanced weights rotated 180^o out of phase. "Resonance" screens have a live frame mounted on a counter vibration screen through inclined leaf springs or rubber buffers. These screens have linear motion and their advantages are low power requirements and that screening can take place on a horizontal deck. Resonance screens were most popular between 1950-1970 but now the trend is towards the two bearing design.[61]

"Multi-angle deck" screens were introduced more than ten years ago by Soprabur. They are designed to provide uniform fines recovery along the length of the screen by varying the angle of the screen deck and thereby the depth of the bed on the screen.[60,62] The deck angle is steepest, 20°-34° at the feed end and shallowest, 5°-10° at the discharge end. Amongst the multi-angle deck screens available are the Pennsylvania Crusher, model BHG, the Krupp model DV and the Soprobur HCC. The BHG is referred to as the "banana screen" because of its appearance, but its capacity is said to be nearly double that of a conventional screen. Multi-angle deck screens are also made with more than one deck.

The Mogensen Sizer is a multi-deck screen supported from above by springs and vibrated by an enclosed motor equipped with unbalanced weights, (Figure 11). The screen motion is linear and the Sizer has three or five equal length screens inclined towards the lower decks at increasingly steep angles.[60] The deck apertures are chosen so that they are larger than the majority of the feed particles. Since the screening surfaces are steeply inclined, the effective aperture openings are reduced and clogging by particles is almost eliminated. This screening method is often referred to as probability screening. As the residence time of material in the Mogensen Sizer is very short, its capacity is high for its size. Units of 0.5 m - 2 m widths are made to screen from 0.1 mm - 100 mm. The capacity of a large unit at 0.5 mm separation size is 1-5t/h while at 10 mm separation size the capacity reaches 80t/h. The decks may be electrically heated to improve screening of moist sticky feeds. In Pennsylvania, Mogensen Sizers have been used successfully for screening anthracite fines.[63]

The Mogensen Divergator is a static or vibrated inclined rod deck for coarse sizing between 25 mm - 40 mm, (Figure 12). The bars are set alternately in two planes so that the spacing between the bars becomes progressively greater,

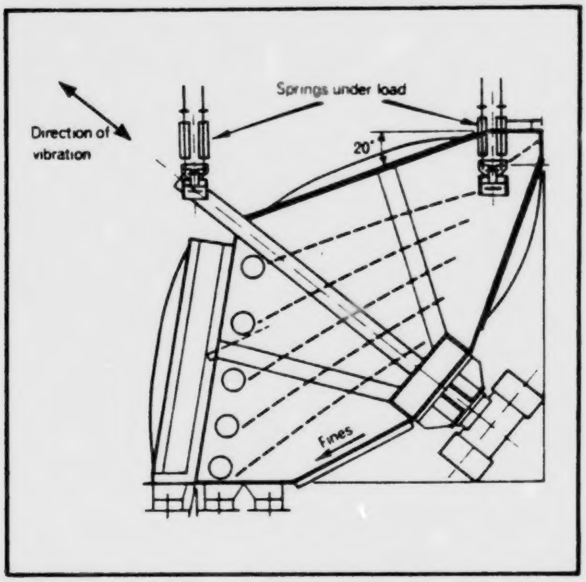

Direction of vibration
Springs under load
20°
Fines

FIGURE 11: Schematic diagram of the Mogensen Sizer

providing a self cleaning action.[64] The capacity of a unit for sizing coal is around 100t/h.

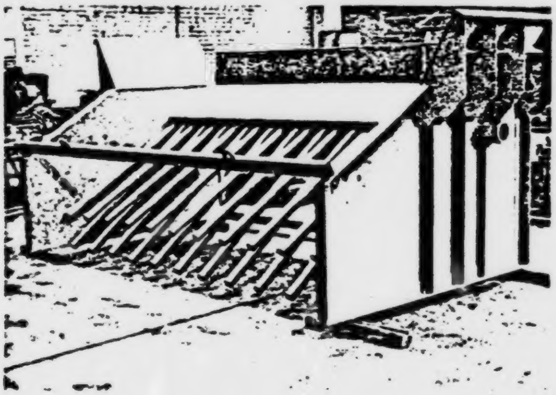


FIGURE 12: The Mogensen Divergator rod deck

The rotating probability screen was developed by the Mining Research and Development Establishment, Bretby, of the National Coal Board, United Kingdon, for the removal of fines from damp raw coal.[65] The main application of the screen is for high moisture content materials in situations where the ability to handle difficult material is more important than accurate size separation, (Figure 13). The screen consists of a circular screen deck horizontally mounted on a vertical rotating shaft driven by a variable speed drive. The screening surface is made from small diameter stainless steel rods radiating from a central hub. The feed material is distributed uniformly from a rotating feed table by a stationary spiral plough mounted above. As feed material is spread over the rotating rod deck finer particles pass between rods while coarser particles are moved to the periphery by centrifugal forces. Both products fall onto a rotating table where they are kept separate by a cylindrical partition and are then discharged by ploughs. The principle of operation is based on the greater probability of finer particles passing through the relatively large apertures between the rotating rods. The effective screening size depends on the rotational speed of the deck which may be regulated either manually or automatically. The screens have 2.4 m diameter decks and a capacity of around 100t/h when set to screen 19 mm - 0 raw coal at 4 mm. Industrial tests carried out at three British collieries have shown the screen to have good reliability, needing little maintenance, and to be capable of accepting raw coal feeds up to 150t/h.[66,67] In order to accept coarser feed material up to 100 mm a scalping deck was fitted to the rotating probability screen. This deck runs at a slower speed and relieves the lower deck of coarse material.[68] A further modification involved strengthening the main bottom sizing deck of the screen. The screen rods were bent down through 90°and their ends were secured in a polyurethane strip. This modification significantly extended screen deck life.[69] The rotating probability screen, known as RoPro is manufactured under licence by Magco Limited. In the U.S.A. it is available through T.J. Gundlach Machine Division, (a Rexnord Company).[70] Apart from coal treatment, interest in the RoPro has been shown by industries including aluminum/bauxite, flour milling, gypsum, rubber reclamation, brewing, sewage, sugar beet and domestic refuse screening.[71] The RoPro was tested for coal fines cleaning by the Fuel Research Institute of South Africa. These tests also confirmed that blinding problems do not arise. Noise and vibration levels were suprisingly low, allowing normal conversation in the immediate vicinity of the screen. The separation efficiency was found to be good.[72] Although this device was specially designed for coal preparation it is reviewed here because it has great performance for fines removal from difficult feeds.

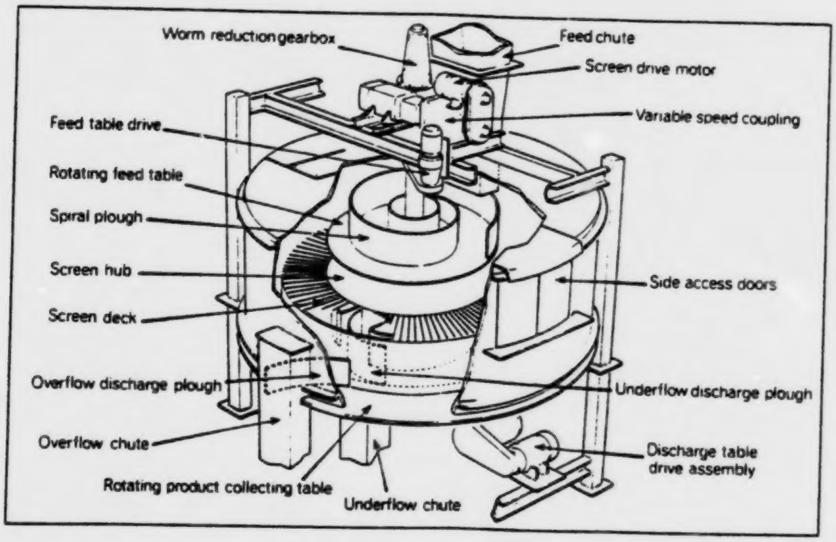

Worm reduction gearbox
Feed chute
Screen drive motor
Variable speed coupling
Feed table drive
Rotating feed table
Spiral plough
Screen hub
Side access doors
Screen deck
Overflow discharge plough
Underflow discharge plough
Overflow chute
Discharge table drive assembly
Rotating product collecting table
Underflow chute

FIGURE 13: The rotating probability screen.

Fine screening gives very low specific throughputs because the capacity per unit area of a screen is approximately proportional to the aperture size. Additional problems in screening fine materials arise from the adhesive forces present between particles themselves, as well as between particles and the screen cloth. Adhesion is caused by thin liquid films and van der Waals and electrostatic forces. Air jet screening is said to overcome these problems.[73] For fine particles, air jet screening capacities may be as much as 100 times those of mechanical screens and are substantially independent of separation size. Specific throughput rates between 30 - 100 t/m^2h are quoted in the size range 20 μm - 100 μm. The pressure of air jets (100 - 300 kPa, 15 - 45 lb/in^2) is able to overcome adhesion forces although some air jet sieves work at much lower pressures. Several machines of different designs exist but they all incorporate a common factor, that the jets of air impinge on the mesh from the underside to prevent blockage. The combination of the air jet principle with mechanical screening has also been used with improved screening performance. In the dry processing of fines, air jet screening offers a sizing technique independent of particle density as compared to dry classification in which density is a significant factor.

Dry screening may be regarded as an ideal sizing operation in which material is screened without the impediment of liquid adhesion.[74] The only disadvantage of dry screening is the dust that it generates. Raw material feeds are generally damp or wet and drying uses considerable amounts of energy and is therefore expensive. It is often technically and economically advantageous to wet screen. In this case the sequence of processing steps is changed from dewatering - drying - dry screening to wet screening - dewatering - drying. The term "wet screening," as normally used, covers the moisture range from a little adhering liquid to complete saturation of the voids between particles of the bulk material. It is more corect to use two terms, "damp screening" and "wet screening," to describe the two different screening actions. In damp screening, size separation is hindered by adhesive forces due to liquid menisci, and therefore this type of process is not very efficient. In wet screening, the feed behaves like a suspension, adhesive forces are negligible and drag forces exerted by the liquid on particles are utilized in order to assist the screening process.

Liquid may occur in feed material in the following forms: (1) internal moisture; (2) pore moisture; (3) surface moisture; (4) intercapillary liquid; (5) cavity liquid. Internal and pore moisture do not cause adhesive forces and so do not hinder screening. Surface moisture tends to collect at contact points between particles and between particles and the screen surface, forming intercapillary liquid in menisci which creates adhesive forces. When sufficient liquid is present (30% to 40% by volume) the three phase boundaries (solid - liquid - air) are eliminated and no capillary forces exist, which is the condition of true wet screening. At intermediate moisture contents of the feed liquid adhesion forces may be significant, particularly for fine particle sizes. For example, for a 1 mm diameter quartz particle the adhesion force may equal 20 times its weight. Although the adhesion force may be greater than the particle weight, screening is still possible, provided that screen acceleration forces are sufficiently high, (Figure 14). Thus the damp screening process may be improved either by increasing screen acceleration or by reducing adhesion forces. The screen acceleration may be increased either by lengthening the amplitude of motion or by raising the frequency of motion. Since screen acceleration is proportional to the square of the frequency and proportional to the amplitude, raising the frequency is a more efficient way of increasing acceleration, but due to the elastic behaviour of moist material an increase in amplitude is also advantageous. Acceleration of up to 20 times gravitational acceleration may be achieved when the vibration is directly applied to the screen cloth by electro-magnetic means. However, electro-magnetic vibrators tend to give small amplitudes. Balls placed under the screen surface also increase screen acceleration, although their main purposes are to reduce screen blinding and to clear adhering moist material. A step incorporated in the screen deck surface aids damp screening as the adhering moist particulate bed has to re-arrange itself on passing over the step.

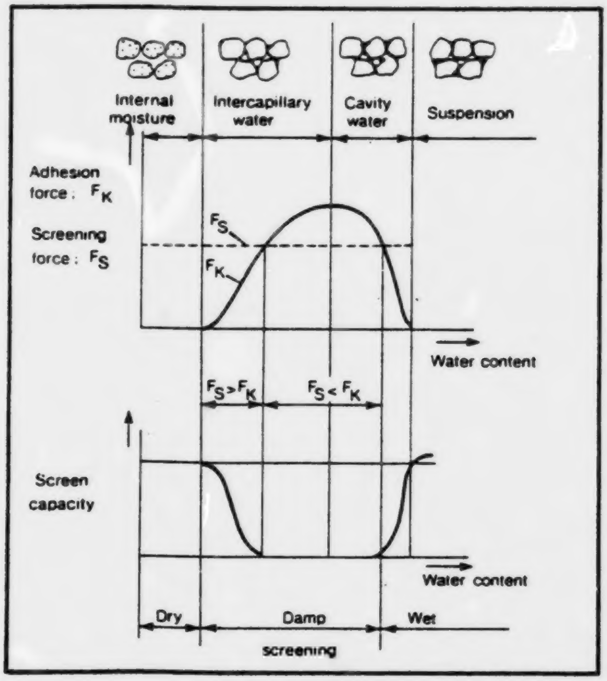


FIGURE 14: The interdependence of adhesion force, screen acceleration force and screening capacity.

The adhesion force is proportional to the surface tension of the liquid and also depends on the contact angle of the liquid with the solids. Heating the screening surface, usually electrically, reduces the surface tension and consequently the adhesion forces and therefore increases screen capacity. Another method of reducing surface tension is by adding surface active agents to the liquid. Various liquid reagents effect the reduction of adhesion forces by increasing the contact angle. Mixing in hygroscopic additives, which absorb moisture, may be very effective in improving screening capacity but may be unacceptable as they contaminate the feed. An example of this is anhydrous lime which binds some of the moisture and then reduces the effective moisture content.

When the feed contains sufficient liquid to exclude air, screening is very favourable on the first contact with the screen cloth. But in a short time dewatering occurs due to screen vibration and conditions of damp screening are created. To maintain wet screening conditions the material on the screen should be sprayed continuously. The Mogensen-Sizer has proved successful in achieving high wet screening efficiency and clog-free operation. When screening limestone-chalk sludge at 0.7 mm a 1.3 m^2 unit has successfully handled 200t/h solids. The multiple inclined decks are arranged so that lower decks have finer apertures and are more inclined. Each deck is sprayed by a number of nozzles and therefore spray water accumulates towards the lower decks, where it is most needed.

Another efficient method of wet screening utilises wedge wire screening surfaces, in the form of a sieve bend, as originally developed by the Dutch State Mines. The sieve bend has a different action from the wet screen. The sieve bend uses the drag forces exerted on particles by the liquid and does not test whether particles are small enough to pass through the slits. Since the silt does not assume the function of a mechanical obstacle for the solid, its width should not be regarded as the separation size. The slit width controls the flow of suspension through the wedge wire screen and the velocity of the flow determines the drag forces on the particles. As the suspension, introduced under pressure, flows downwards on the sieve bend, smaller particles pass through the screen due to the liquid drag. Larger particles, with greater inertia, are not diverted from their paths by liquid drag forces and continue to glide parallel to the screen surface and then report to the oversize product. The gliding action reduces the abrasion on the wedge wires while the constant flow of relatively high velocity reduces the build-up of solids and foreign bodies, such as fibrous materials, on the screen surface. Based on the original Fontein patent, Dorr-Oliver manufacturers a whole range of DSM sieve bends. The range covers bends with 45° to 320° arc angles and 0.05 mm to 4.0 mm slit widths. The throughput capacities of the sieve bends are high, in between 40 and 300 m^3/h for every square metre of screen surface.

The Bauer-Hydrasieve manufactured by Krahn-Chemie is said to be an improved form of the DSM sieve bend, from which it differs in two ways. The wedge wires used have rounded corners which modify the pulp flow through the slits (Figure 15). Conventional sharp-edged wedge wires cause abrupt change in the flow by diverting part of it, but when the wedge wire corners are rounded the fluid film flows smoothly following the rounded trailing edge of the wedge wire (Coanda effect). This liquid film motion leads to clear separation of even fine particles with relatively small shearing forces. The other difference in the Hydrasieve is that the wedge wires are not straight but are made to form waves within the screen surface. The wave form is said to ensure that the screen does not become clogged, even in critical areas such as the vicinity of the supporting wires.

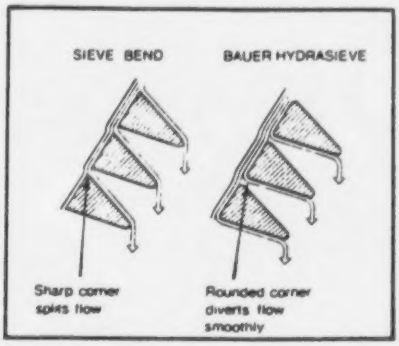


FIGURE 15: Different flow patterns through a standard sieve bend and through the Bauer Hydrasieve.

The Wallis Ultrasonic Separator was tested at the De Beers Diamond Research Laboratory, and was found to screen sizes below 750 µm very efficiently.[75] The separator consists of a 0.77 m wide screen deck mounted at an angle of 35° to the horizontal below which a sealed ultrasonic transducer is positioned. (Figure 16). The transducer is powered by a 2kW ultrasonic generator operating at 18kHz. The feed pulp is introduced into a feed box at the upper end of the screen. The pulp water is used both to cool the transducer and for the transmission of ultrasonic waves. Some unexpected results have been obtained during testing. The efficiency of the separator appears to increase for finer screen apertures. It was found that the ultrasonics had practically no effect on the screening efficiency but only prevented blinding of the screen. The Wallis Separator operates at relatively high feed rates; in one test very good separation was obtained at a solids feed rate of approximately 15 tonnes/h at 104 μm separation size with about 35% undersize material. The moisture content of oversize is relatively low, between 16 and 22%. One disadvantage still to be overcome is the short life of the screen cloth.

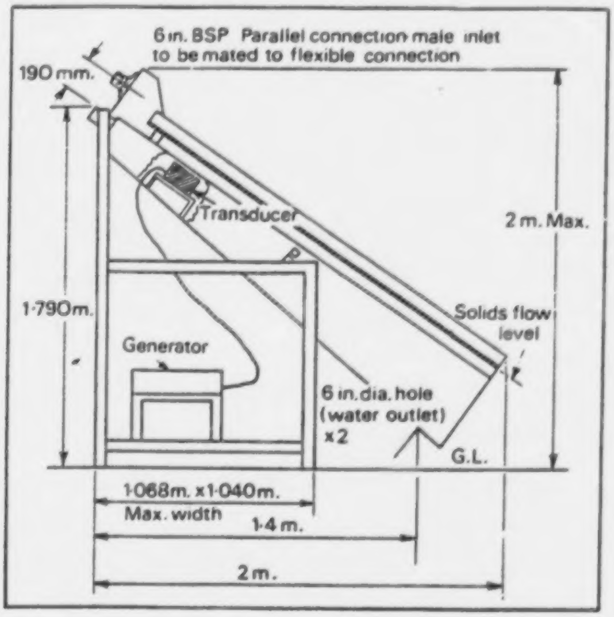

6 in. BSP Parallel connection male inlet to be mated to flexible connection
190 mm.
Transducer
2 m. Max.
1·790m.
Generator
Solids flow level
6 in. dia. hole (water outlet) x2
G.L.
1·068m. x 1·040m.
Max. width
1·4 m.
2 m.

FIGURE 16: Schematic diagram of the Wallis Ultrasonic Separator

Static screens such as the DSM sieve bend and its later version, the Dorr Rapifine have gained wide acceptance in the American taconite industry, in bauxite grinding circuits and in the raw cement and coal industries.[76] The Dorr Rapifine screen was designed to overcome the problem of abrasion by using lower pulp velocities. A periodic rapping is applied to the screen to dislodge adhering particles. The separation size is about half the size of the slot openings. Screen capacities are up to 180 m^3/h for every square metre of screen area and single units are built to handle 450 m^3/h. While the DSM sieve bends have a screen open area of approximately 5% the Bartles CTS screen, which consists of curved stainless steel woven wire cloth has a much greater open area. The unit has a primary screen stage, followed by a scavenging stage. A series of crimps on the back of the cloth serves to divert pulp clinging to the back of the cloth. A later version uses bonded plastic strip for this purpose. Capacity at a separation size of 150 μm is about 26 m^3/h for 1 m width of screen.

Pritchard, of Allis-Chalmers Australia Limited, discusses in two articles considerations in the choice of vibrating screens. The first is concerned with the influence of amplitude and speed of vibration, merits of wire cloth, rubber and polyurethane screening surfaces and means of reducing pegging and blinding.[77] The second paper deals with the sizing of vibrating screens for a given duty by correcting the specific screen capacity at a nominal separation size by means of the various correction factors.[78] He also discusses the correct interpretation of screening efficiency and explains how wet screening can overcome some problems associated with dry screening.

A semi-empirical approach, which uses computer model simulation of screening, was developed jointly by Allis-Chalmers Corporation and the U.S. Bureau of Mines.[79,80] The tests were carried out on full size screens with the deck areas reduced to 0.304 m wide by 3.66 m long. The two screens used were both Allis-Chalmers 1.83 m by 4.88 m, one Low-Head with 45^0 straight line motion and the other Ripl-Flo with circular motion. The simulation is semi-empirical using mathematical expressions developed from theory to describe the probability of passage of a certain sized particle, the number of particle approaches during screening and the effect of bed depth on screening performance together with data obtained from screening tests. Within the range of collected data, the model is capable of predicting screen performance. As all variables could not be expressed mathematically these still have to be incorporated in the model as correction factors. Further information on selection and sizing of screens is detailed in text books.[30,50]

2.3 Classification

In classification particles of various sizes, densities and shapes are separated by being allowed to settle in a fluid. The coarse, denser and rounder particles settle faster than finer, lower density and more angular particles. Normally fluid motion carries away the slower settling particles while the faster settling particles are removed as a sediment. Water or air is used as the sedimentation medium. Separation size is usually between 2 mm - 1 μm. Classification is a very important operation in mineral processing and its main applications are:

1. Classification of the feed before treatment on a flowing film concentrator, (e.g., shaking table).

2. Separation of fine valuable mineral from coarse waste, (e.g., clay beneficiation).

3. Separation of coarse valuable mineral from fine waste, (e.g., removal of slime gangue minerals in iron ore concentration).

4. Sizing of fine feeds for which industrial screening is inefficient and costly, (e.g., classifier operating in closed circuit with a ball mill). This operation is not true sizing as particle density and shape also influence separation.

Classification has not been important in coal preparation because coal has mainly been treated at coarser sizes. It is expected that with the increased importance of fines, classification will become a significant part of coal cleaning technology. The principles involved in classification are quite universal as they also affect most gravity processes and are especially important in jig washing, pneumatic and fluidized bed separations. Therefore, the application of classification principles to coal preparation will be briefly outlined.

Particles which have the same settling velocity (equal settling particles) cannot be separated by classification. Thus a larger, low density particle will settle at the same speed as a smaller, dense particle. The settling ratio is the diameter ratio of equal settling particles of different densities. This ratio sets the theoretical upper limit for the size range of the mineral mixture which may be separated completely according to densities, (assuming an ideal separator). When the size ratio is greater than the settling ratio, one or both of the products will have misplaced particles of the wrong density.

The settling ratio is:

$$\left(\frac{d_B}{d_A} = \frac{\rho A - \rho M}{\rho B \quad \rho M} \right)^S$$

where d = diameter of particles
ρ = density
subscript A refers to denser mineral
subscript B refers to less dense mineral
subscript M refers to medium.

S = 0.5 in streamline flow
S = 1.0 in turbulent flow.

When the solids concentration of the separating slurry is very low, the operating density of the medium is that of the liquid and the condition is called "free settling."

At higher solids concentrations the effective density of the medium is the mean density of the coal/shale/water suspension. Increased medium density increases the settling ratio. This is important because operation at high solids concentrations, that is under "hindered settling" conditions, permits separation of wide size ranges according to density. When the density of the medium becomes equal to the density of the less dense mineral, the settling ratio becomes infinity and separation is independent of particle size. Theoretically, this is the case in dense medium separation, where the density of the medium is greater than that of the less dense mineral and consequently the mineral floats irrespective of its size, while the dense mineral sinks.

The settling ratios against the percentage of solids by volume present in the slurry are shown in Figure 17 for coal/shale/water and galena/quartz/water systems. The values plotted are as given by Gaudin in Principles of Mineral Dressing, tables 28, 29 and 30.[14] He assumed that the respective densities of coal, shale, galena, quartz and water were 1.3, 2.5, 7.5, 2.65 and 1.00 g/cm^3. The graphs were plotted for 0%, 25% and 75% of the denser mineral contents. Figures 17a and 17b refer to streamline flow while 17c and 17d refer to turbulent flow conditions. Figure 17c also shows the settling ratio of coal/shale mixtures in air.

It is apparent from these graphs that the density of suspension increases linearly with the volumetric percentage of solids in suspension and that the increasing suspension density sharply increases the settling ratio, and therefore, the separable size range. Furthermore, an increased percentage of the dense component also significantly increases settling ratios through increased medium density. The lower

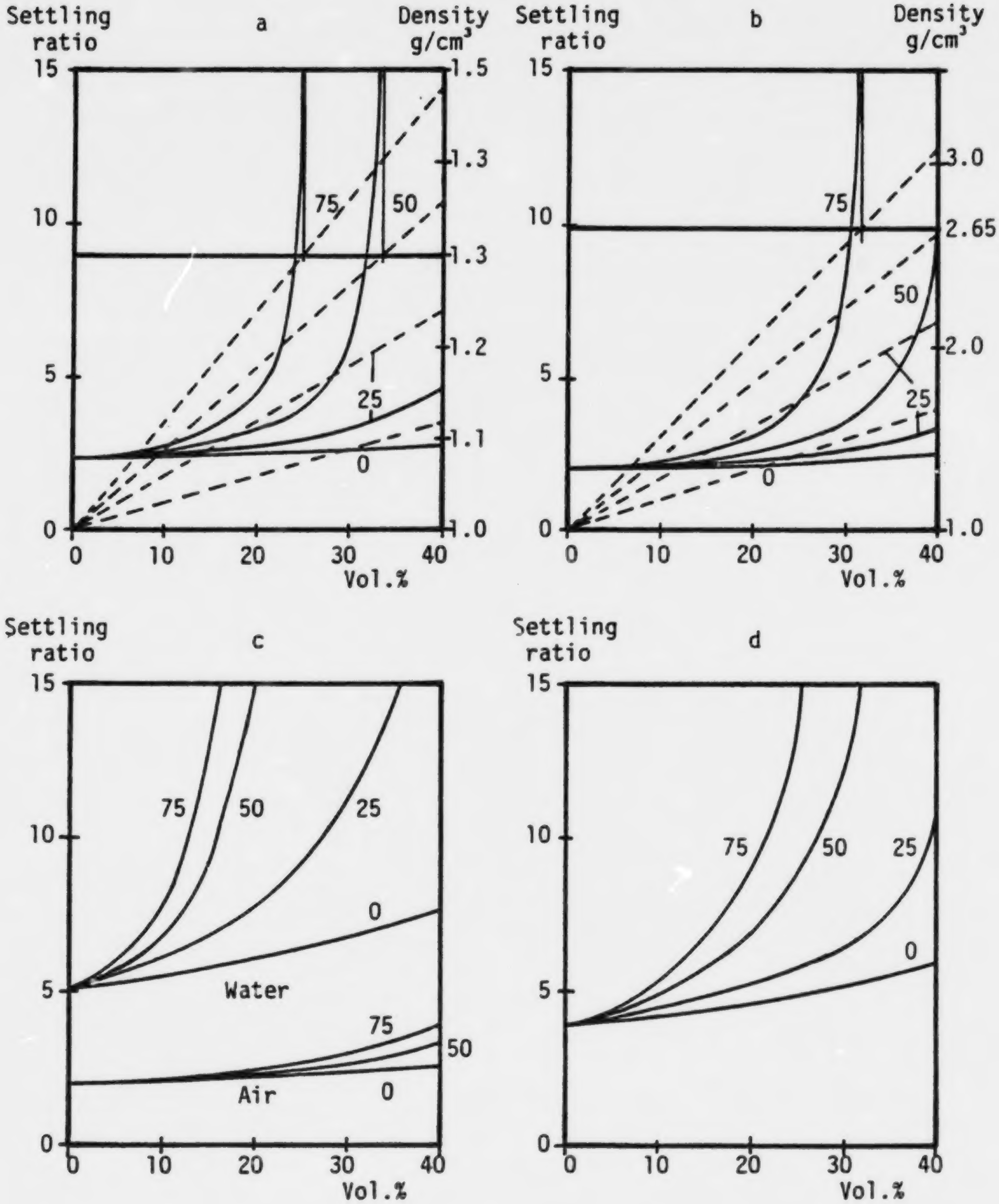


FIGURE 17: Settling ratios and suspension densities against percentage of solids by volume at various percentages of denser mineral content.

a. Coal/shale/water system under streamline flow conditions.
b. Galena/quartz/water system under streamline flow conditions.
c. Coal/shale/water and coal/shale/air systems under turbulent flow conditions.
d. Galena/quartz/water system under turbulent flow conditions.

mineral density is also marked on the graph. The percentages of solids which give this density coincide with the infinite values of the settling ratios.

On comparing these graphs it is evident that settling ratios of coal/shale are as high or higher than for one of the most easily separable mineral combinations. This would suggest that classification (or jigging) of raw coal is simpler than that of an ideal mineral mixture such as galena/quartz. However, shale's lower density compared with that of galena results in lower separation velocities of shale and a consequent lower throughput in the preparation of coal, considering similar particle sizes.

Assuming a coal/shale slurry of 30% solids by volume, containing 25% shale, the density of the slurry is 1.18 g/cm^3. For a galena/quartz suspension of 30% solids and 25% galena content the slurry density is 1.86 g/cm^3. Therefore, the driving force of a shale particle of 2.5 g/cm^3 density in a 1.18 g/cm^3 slurry will be proportional to the density difference, that is proportional to 1.32. Similarly, for a galena particle of 7.5 g/cm^3 in a slurry of 1.86 g/cm^3 density, the driving force is porportional to 5.64. Therefore, the driving force on a galena particle is 4.3 times that on a shale particle of the same size.

The sedimentation velocities of coal/shale systems are only 36% in the streamline flow region and about 67% in a turbulent flow region, compared to the sedimentation velocities of a normal mineral system. The throughput capacities of classifiers are likely to be reduced in similar proportions. There is also a reduction of specific throughput due to the lower density or greater bulk of raw coal. Thus, it appears that the throughput of classifiers operating with coal will be only 18% under streamline flow conditions and 33% under turbulent flow conditions of the capacities of mineral classifiers working in similar size ranges with normal mineral combinations. But, usually the coarser size of coal can compensate for its lower density, as in streamline flow raw coal particles two and a half times the size of mineral particles, and in turbulent flow coal particles nine times as large will give similar capacities. The above considerations also apply to jig washing.

So far it has been assumed that the mineral slurry behaves as a Newtonian fluid. The presence of fine particles usually causes slurry behaviour to be non-Newtonian, with a definite yield strength. In Newtonian fluids there is particle displacement while any driving force is present, irrespective of its magnitude. In a slurry with yield strength the driving force has to overcome the yield strength for particle displacement to occur. Therefore, the reduced driving force of coal/shale systems may seriously affect the classification and gravity separation of fines.

Mechanical classifiers are still used in mineral processing but centrifugal classification using hydrocyclones is now the most popular method and centrifuges have also become accepted. Mechanical classifiers have had minor use in coal preparation, although spiral classifiers are sometimes used for magnetic medium densification. The use of classifier hydrocyclones is accepted for coal fines removal and water clarification. It is expected that classifier hydrocyclones will be used to greater extent in coal fines treatment, including coal grinding circuits.

Hydrocyclones are simple and versatile devices. It is unfortunate that the separation size of a hydrocyclone is closely related to its diameter. Thus small hydrocyclones with low capacities must be used for fine classification and large diameter hydrocyclones with very high capacities are only suitable for coarser separations. The upper limit of their useful separation size is normally around 250 μm with mineral feeds and somewhat higher for coal. The sizing and selection of hydrocyclones for classification purposes is given in reference 30, chapter 32 and reference 50, chapter 17.

Although the construction of hydrocyclones is simple, it is not possible to predict their performance theoretically from their operating and design variables. Many empirical relationships have been developed for the prediction of cyclone performance.

Hydrocyclone throughput and separation size are the most common variables which are related to cyclone dimensions and operating variables.[81,82] Lynch gives separation size formulae for hydrocyclones in which the constants have to be determined from tests.[31] Plitt gives empirical relationships for separation size, volumetric split between overflow and underflow, volumetric throughput and sharpness of separation in terms of operating and design variables. In his formulae the empirical constants are given. The prediction accuracy of the Lynch and Plitt models was tested on a large scale grinding process at a lead-zinc concentrator. This study found that the Lynch model gave acceptable predictions at low feed rate conditions but poorer predictions at higher feed rates. Plitt's equation did not fit well when his constants were used but they agreed well with new constants. Thus it seems that for adequate industrial modelling, it is advisable to determine the constants in the empirical relationships for each operation.[84]

Operating difficulties with small hydrocyclones often occur through the blockage of their outlets. Such blockage may even occur when used in conjunction with efficient screening. The blockage may be caused by scale from pumps or pipes or by fibrous material which passes through fine screen openings and then coils up to block a relatively large hole. Richard Mozley Limited is marketing a cyclone system which overcomes most of these defects.[85] The cyclones, moulded from cheap abrasion-resistant plastic are grouped together and up to forty cyclones may share one feed inlet and one outlet for each of the two products. These cyclones are available with either manual or automatic cleaning. Blockages are cleared by pushing clearing rods through the cyclones from the vortex finder end. The automatic version has a thermistor sensor at each apex discharge to monitor the discharge and to initiate instant clearance. The hydrocyclone system offered can classify, degrit, deslime and dewater in the 1 μm - 20 μm range and is considered to be a viable alternative to the more expensive solid bowl centrifuge for many applications.

There is a considerable variety of hydrocyclones available worldwide with widely varying specifications and with diameters from 10 mm to 1250 mm. An international guide to hydrocyclones lists seventeen manufacturers and the cyclone diameters they offer.[86] Brief descriptions of the intended uses and some construction details are included together with the addresses of manufacturers.

The choking-free cyclone was developed in Japan for the removal of fibrous materials from municipal waste, but it was found to have other applications such as desliming fine sands.[87] This non-choking cyclone is basically an ordinary hydrocyclone equipped with a screw conveyor at the apex discharge end, (Figure 18). The screw has a hollow shaft which allows the entry of air to maintain the air core. The speed of the screw conveyor is variable and may be set to suit separation conditions. The classification efficiency is similar to that of a conventional cyclone. In tests on a 320 mm dia. unit of 10° apex angle, treating slurry at 30 to 120 m^3/h, it was found that operation was stable, even at high underflow solids concentrations. The separation size is said to be variable between 20 μm - 4 mm simply by regulating the rotational speed of the screw. The classification performance of this cyclone is closely related to the torque of the screw conveyor and controlling the speed in relation to torque makes it possible to operate this cyclone very efficiently.

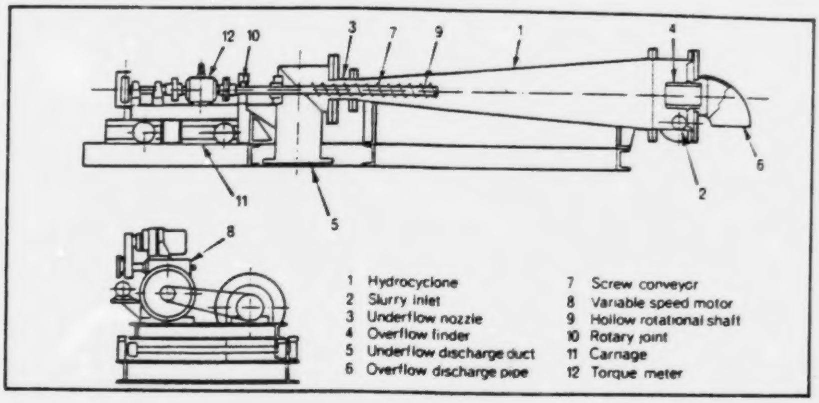

12
10
3
7
9
1
4
6
2
11
5
8
1 Hydrocyclone
2 Slurry inlet
3 Underflow nozzle
4 Overflow finder
5 Underflow discharge duct
6 Overflow discharge pipe
7 Screw conveyor
8 Variable speed motor
9 Hollow rotational shaft
10 Rotary joint
11 Carriage
12 Torque meter

FIGURE 18: Schematic diagram of the choking-free cyclone.

The Mining Research and Development Establishment of the National Coal Board, England, has developed an efficient and low cost hydrocyclone filter.[88] The device is primarily intended to prevent blockages of water sprays by removing solid particles from the water. As shown in Figure 19 the water is injected tangentially into the hydrocyclone filter and leaves through an axial outlet at the top. The solid particles collect in the collection pot and may be discharged through an outlet valve at the bottom of the unit. The extraction efficiency at 190 μm particle size is stated to be 95%. The 100 mm internal diameter hydrocyclone filter will pass 160 l/min of water at 100 kPa (14.5 lb/in^2) pressure drop.

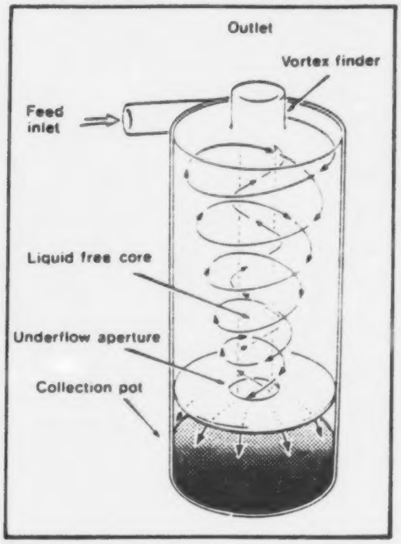


FIGURE 19: Hydrocyclone filter unit.

The blind cyclone, from Richard Mozley Limited, is not a separator but is a means of reducing water flow of jig hutch products without risk of blockage by large particles. It is a short hydrocyclone without a vortex finder. In cyclones a back pressure is developed, due to centrifugal forces, which can be used to reduce flow rates. The principal has been used on several types of separators such as jigs and the Vorsyl separators where it is referred to as Vortextractor. The 5" blind cyclone is said to reduce the flow rate to 25% of its normal value.

Classifier hydrocyclones, operated over 10 μm separation size have the advantages of low operating costs, small space requirements, good capacities and simple design. For finer separation sizes below 10 μm, hydrocyclone assemblies containing many cyclones are necessary to maintain reasonable throughputs and this makes the installation less compact, more complex and more expensive. The accuracy of classification is also reduced at finer separation sizes. Therefore in some circumstances use of solid bowl centrifuges may be preferred.[89] Solid bowl centrifuges are capable of producing reasonably sharp cuts down to 3 μm, but the capital cost and operating costs are usually higher than those of hydrocyclones. For the classification of phosphate by two stage solid bowl centrifuging, the operating cost was estimated in 1980 as US $17.20/t of solids treated.

According to Professor Hukki of Finland, the efficiency of classification in a closed grinding circuit is very important. For this reason he advocates the use of two stage classification as he considers that hydrocyclones alone do not give sharp enough separation.[90,91] The proposed two stage classification would use hydrocyclones for the first stage and the cyclone underflow would be further classified by a hydraulic cone classifier of Hukki's own design. Efficient two stage classification is reported to have the following advantages:

1. Increased sharpness of size separation permitting a 25% increase in grinding capacity.

2. Reduction of energy consumption by up to 20%.

3. Single stage closed circuit rod milling possible.

4. Reduction of specific surface area of ground product requiring less flotation reagent and giving increased flotation rates.

5. With lower fines content of the ground product there is no need for desliming, the recovery and concentrate grade are increased, thickening, filtration and drying are easier.

The two stage classification system originated by Hukki has been tested in three of Outokumpu Oy's concentrators, Virtasalmi, Hammaslahti and Vuonos.[92] In these installations the fines recoveries of the hydrocyclone classifiers were between 50% - 60%. Following the installation of Larox cone crushers, fines recoveries increased to 70% - 85%, with around 40% reduction in circulating loads. Test results at these concentrators showed throughput increases between 4% - 14% with narrower size distribution of the ground product and an increase in flotation recovery. Two stage classification may be advantageously applicable in coal grinding circuits to increase grinding efficiency and reduce fines content of the ground product. The lower ultrafines content may be of importance in reducing the viscosity of coal-liquid mixtures and Carbogel.

When grinding in closed circuit with a classifier the denser mineral is preferentially returned to the grinding mill for further grinding. This is because in a classifier, which is not a true sizing device, not only the size but also the density governs separation. The fact that the denser particle is returned to the mill at finer sizes than the less dense results in overgrinding of the dense mineral (usually the useful mineral), which is undesirable as it leads to lower recoveries due to slimes losses. However, the same effect in grinding coal becomes an advantage, as the shale or pyrite containing dense middling particles are returned for further grinding. Considering the hindered settling ratios of particles, one finds that a locked particle containing 20% pyrite by volume and 80%

coal will be returned to the mill even when its size is half or one third of the separation size of the classifier. This results in higher pyrite liberation without additional grinding of the coal. The above described preferential grinding of dense minerals also has implications in laboratory testing of pyrite liberation. If a laboratory batch rod mill is used for grinding coal in pyrite liberation studies the results should not be expected to simulate industrial closed circuit grinding. The pyrite liberation in closed circuit grinding will be greater than in batch rod milling at the same overall size distribution. Finch and co-workers of McGill University have shown that considerable differences exist between individual mineral size distributions in laboratory batch and closed circuit grinding of a lead-zinc ore.[93] When predicting performance of plant operation from laboratory tests the different size distributions of individual minerals in the ground products should always be considered and not only the overall size distribution of the ground product. With dense useful minerals the predictions tend to be too optimistic, while in pyritic sulphur liberation the predictions tend to be low.

So far wet classification appliances have been considered but classification may also be carried out with air as the sedimentation medium. Machines which use air are termed pneumatic classifiers. Less technical information about the performance of pneumatic classifiers is available than for wet classifying equipment. A new static pneumatic classifier for fine classification in grinding circuits was primarily developed for cement production and is said to be able to handle up to 100t/h.[94] The classifier has no internal moving parts and a conventional low or medium pressure blower provides the energy for the separation, (Figure 20).

Centrifugal forces are used in the classifier to separate coarse particles from finer. The design of the unit has been optimized for minimum flow resistance and minimum flow turbulences in order to maximise sharpness of classification and overall performance and to minimize overall energy consumption.

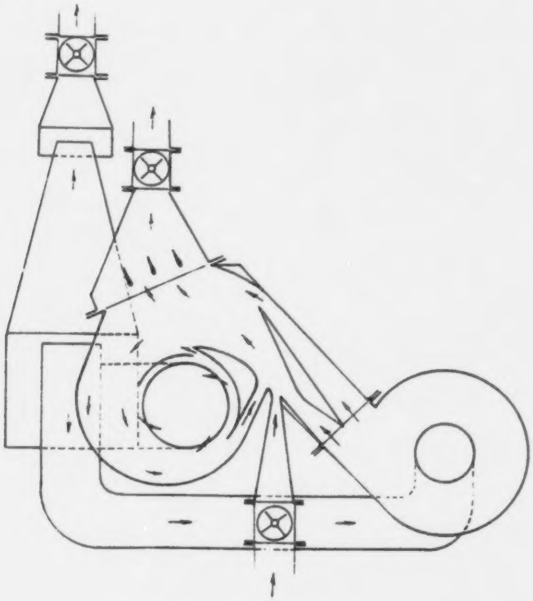


FIGURE 20: Schematic diagram of the static centrifugal classifier

A new centrifugal deduster for removing the bulk of the minus 0.5 mm dry fines before wet treatment is being successfully used at Lady Jane coal preparation plant, Pennsylvania.[95] This deduster reduces wet slurry treatment and decreases the overall moisture content of the clean coal. The coal treated is from the Lower Kittaning Seam and is a medium volatile coal of high calorific value and of high pyritic sulphur content. The coal is crushed to below 19 mm to liberate some pyritic sulphur. The crushed product is fed into the centrifugal deduster which removes up to 80% of the minus 0.5 mm fines, depending on the moisture content of the feed. With a total moisture content of 6% the removal is only about 50%. Rotating at 140 rpm and fed at 230 t/h the deduster removed 11.2% of the feed as fines (Figure 21). The high ash and high pyritic sulphur fines tend to remain in the coarse product which is an advantage as the coarse product is treated in a Batac jig and rejected at that stage. The 4.7 m diameter centrifugal deduster requires 110 kW for fan drive, 37 kW for the distributor plates and 15 kW for the dust discharger. This deduster has been used successfully at the Lady Jane coal preparation plant and also in various German coal preparation plants and its operation has been virtually troublefree. More general application of centrifugal dedusters in coal preparation plants should be considered since they minimize the costly wet slurry treatment and also reduce the moisture content of the clean coal.

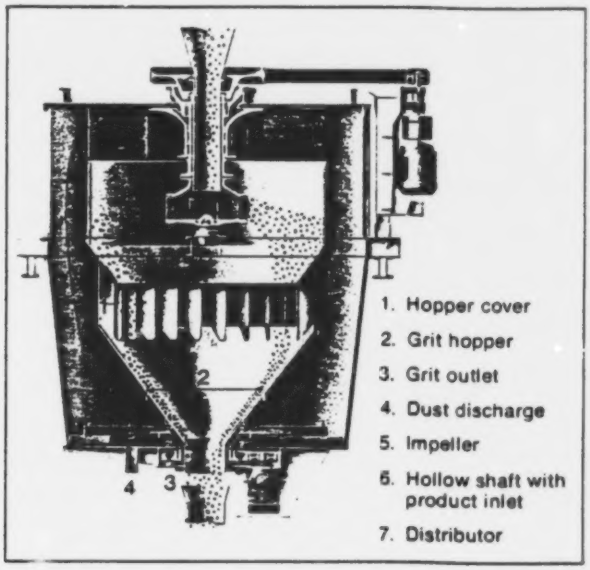

2
4
3
1. Hopper cover
2. Grit hopper
3. Grit outlet
4. Dust discharge
5. Impeller
6. Hollow shaft with product inlet
7. Distributor

FIGURE 21: Cross-section of the centrifugal deduster

2.4 Gravity separation

Gravity separation was the mainstay of mineral processing and coal preparation until the advent of flotation in the 1920's, after which flotation gained ground in mineral processing. In coal preparation, gravity separation, mainly in the form of jig washing, remained dominant. Gravity separation is a relatively cheap process which requires no reagents and thus is an environmentally more acceptable process. On coarse feeds, gravity separators have large throughput capacities which diminish towards the finer sizes. In 1977 in the U.S., 313 Mt of feed was processed by gravity concentration including 228 Mt of coal and in the same year approximately 400 Mt of feed was treated by flotation, according to Mills.[16] Gravity separation appears to be making a comeback. While froth flotation is a very versatile process its applicability is limited by the restricted upper size limit of feed, by high reagent costs and also to some extend by the way it responds to middling particles. In gravity separation there are many types of separators operating on slightly different separating principles, thus offering varying separating characteristics. The great variety of gravity separators offers innovative applications. Perhaps coal preparation plant designers should consider and evaluate the applicability and cost efficiency of some of the less frequently used gravity separators to exploit their capabilities.

It is virtually impossible to develop quantitative theoretical principles for gravity separation processes. The characterization of particle size, particle shape and density distribution of feed (all three are involved in governing gravity separation), is formidable and there are many additional factors affecting the separation process.

However, it is possible to formulate a semi-quantitative model which is useful in explaining the main tendencies in separation. The more general exploitation of gravity separation is impeded by the complexities involved in its separation mechanism, which are not widely understood as few operators would consider themselves as experts in the whole of this field. Mills gives a good summary of the theory of gravity separation.[97] He considers jigging, the application of flowing film concentration theory to shaking tables, spirals, cones and sluices.

2.4.1 Jigging

In jigging, the mixed mineral bed experiences water pulsation which repeatedly fluidizes the bed, allowing denser particles to stratify in lower layers, while lower density particles take up positions in the upper levels. It can be shown that the stratified bed is in a lower potential energy condition than the mixed bed and this lowering of free energy causes the stratification. The jigging cycle is needed to repeatedly remove interparticle friction and to allow particles to separate into stratified layers. According to Gaudin, (chapter 12), three main factors contribute to stratification in a jig.[14]

1. Hindered settling. When separation occurs according to terminal velocities under hindered settling conditions. Particles usually reach 99% of their terminal velocities after a few milliseconds sedimentation time after which they settle at substantially constant velocities. Large, dense particles sediment fastest, fine, low density particles settle slowest while smaller, dense and larger, less dense particles have similar settling velocities. This causes stratification according to sedimentation velocities.

2. Differential acceleration. In any jigging cycle, when the relative velocity of particle to water is very low there are no fluid drag forces and the particles have a maximum acceleration of

$$\frac{\rho_s - \rho}{\rho_s} g$$

where ρ_s and ρ are densities of solids and medium and g is acceleration due to gravity. Although the differential acceleration period is short, it is useful, as during this time separation is independent of size and only according to density. During the acceleration period shale accelerates 2.6 times faster than coal, even in free settling conditions and higher in hindered settling.

3. Consolidation trickling. Due to hindered settling the finer particles of a mineral stratify in higher levels than coarser particles of the same mineral. As the upward current of the pulse dies away the coarser particles consolidate to a porous bed. Finer particles which remain longer in suspension, as they have lower settling velocities, work their way down through the voids between large particles. Thus the mineral bed tends to become a mixture of coarser and finer particles due to consolidation trickling.

The three effects together are responsible for the final stratification of the bed. The relative importance of the three factors greatly depends on the type of jig and its operating conditions. The importance of differential acceleration is probably negligible in fast jigging cycles applied to fine particles. Consolidation trickling is more pronounced in jigs with suction pulsation.

The pulsation may be created either by a plunger, diaphragm, water pulsation, air pulsation or by movement of the screen supporting the bed. For each type of pulsation mechanism there are different jigs available. In coal preparation, air pulsated jigs have been the most popular.

The removal of the stratified components of the bed into low density and high density products is often achieved through a gate and dam discharge. The recommended thickness for a stratified bed is normally 7-15 particle diameters for coarse jigging and 15-30 particle diameters for fine jigging. At the discharge end the dense product is in hydrostatic equilibrium with the whole stratified bed and thus the discharge is self-controlled. Shale discharge is often assisted due to its coarse size and lower density. The above method is known as jigging on the screen. An alternative method is jigging through the screen, where the stratified dense mineral is removed through the screen as a hutch product. To prevent the passage of low density minerals through the screen, a dense bedding or ragging which is coarser than the screen is used. For dense minerals the ragging may be steel shot, while for coal, felspar is commonly used. Jigging through the screen is advantageous for the finer particle size range as the well chosen ragging material selectively prevents the passage of lower density material which results in sharper separation. The gradual removal of dense stratified layers also reduces the separation distance and stratified bed depth thereby increasing throughput capacities. With the increasing fines proportion of coal, it is expected that the importance of jigging through the screen will continue to rise.

In jigging the denser component stratifies in the lower layers, in reverse stratification the opposite occurs. When particulate mixtures of different sizes and densities are vibrated in water or air with sufficient vibration energies, and at frequencies between 2-26 Hz, stratification takes place in which the denser component appears in the upper layers. It is usually sufficient to subject the system to vibrations for 15-30 seconds. This may be caused by the denser particles possessing greater vibration energies which cause them to float upwards. According to investigators the phenomenon could be utilized for the treatment of coal fines. The water requirements would be half to one third of those required for jigging and the capacity of the process is predicted to be 7-10t/h for every square metre of separator area. Reverse stratification may be worthy of further investigation for the wet or dry treatment of coal fines. Reverse stratification should also be considered for its effect on jigging, as intense pulsation may create partially reversed stratification and may counteract useful separation to some extent.

The Batac jig was developed by Humboldt-Wedag of Germany for coal washing. The name is derived from Baum and Tacub, the latter being a jig made in Japan. Although it is not used in mineral processing, a brief summary is considered worthwhile. The Batac jig is pneumatically operated but has no side air chamber like the Baum jig.[98] Instead it is designed with a series of multiple air chambers extending under the jig for its full width, thus giving uniform air distribution. The jig uses electronically controlled air valves which provide a sharp cut off of the air input and exhaust. Batac jigs are made for cleaning coarse coal but they are also effective in fine coal cleaning using a felspar bed.[99,100] Apart from those in Germany, at least five coal preparation plants in the U.S. operate with Batac jigs.[101] For 19 mm - 0.5 mm feed the probable errors of separation are usually below 0.1 g/cm^3 and for separating 0.5 mm - 150 μm coal fines, probable errors of less than 0.2 g/cm^3 are normal. Low volatile coals tend to be washed at greater efficiencies than high volatile coals. The width of the jig is up to 6 m resulting in capacities of 1000t/h for 10 mm - 120 mm feed and 600t/h for less than 10 mm feed.[102,103]

Humboldt-Wedag has introduced a jig specially developed for separation of fine coal below 3 mm.[104] It has been realised that by dividing the feed into two size fractions which are jigged separately, better results are obtainable than when jigging a wide size range. The fine coal jig is an electronically controlled felspar jig. It is made up from the jig compartment and side mounted air compartment, (Figure 22). The jig bed is 2 m long and 1 m wide. The feed enters opposite the air chamber and the clean coal is discharged into a launder near the chamber. The bed is carried on a 5 mm square mesh perforated sheet which also carries the 6 mm - 50 mm felspar ragging. Stratification in a relatively shallow bed is fast so capacities up to 5-6 t/m^{2}h are reported. The pulsation is adjustable between 30-170/min. The results of a three week continuous test on finer than 3 mm coal feed at 9.5t/h and 120/min pulsation rate showed that it reduced the feed ash from 13.4% - 5.3% in the clean coal and produced 80.4% refuse ash. The separation density, depending on the particle size, varied from 1.67 g/cm^3 - 2.04 g/cm^3. The total yield of clean coal was 92.6%. One area where the fine coal jig is of particular advantage is in coal preparation plants operating with wet prescreening, for example, before dense medium separation. The normal practice of wet screening at 0.5 mm involves large screening surfaces with the associated wear problems. Desliming at around 2 mm and treating the finer than 2 mm fraction in a fine coal jig not only simplifies the wet screening problem but relieves the dense medium separation of the material which is difficult to handle. As a result, the overall feed rate to the washery may be increased and dense medium losses reduced. Another advantage is the improved pyritic sulphur removal by the fine coal jig. It appears that the incorporation of fine coal jigs in coal preparation plants, whether they employ jig washing or dense medium separation, should result in advantages with regard to improved overall performance and pyritic sulphur reduction. The dewatering of fine clean coal can be effectively carried out in either pusher or solid bowl centrifuges.

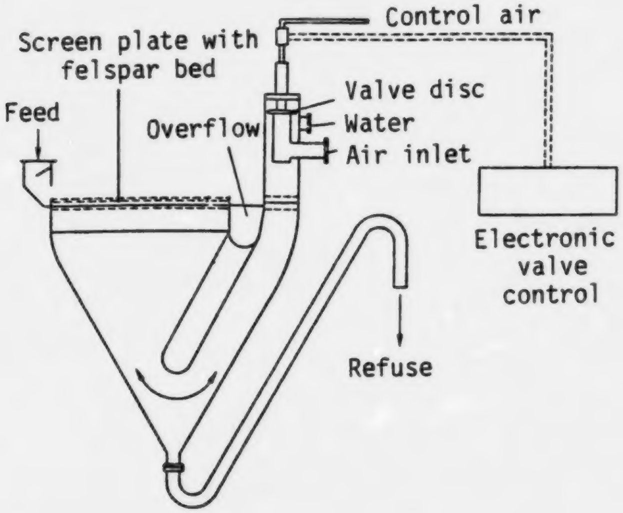


FIGURE 22: Schematic diagram of a fine coal jig.

Baum jigs have been in use for over eighty years and they have been especially popular in Europe.[105] Various improvements have been made, especially to ensure controlled removal of refuse. One modification devised by Wallace was radar operated shale extraction (ROSE) in which an open ended tube is attached to the wash box below screen level in each of the shale discharge compartments.[106] The water moves in the tube with each pulsation and the water height varies proportionately to the weight of shale in the bed. The water level is continuously measured by a radar pulse echo system and the signal it emits controls the shale extraction rate. The jig has a special compartment in which pulsations are generated solely to discharge the shale without disturbing the stratified raw coal.

Another system for shale extraction employs a pressure transducer to sense the shale bed height and to electropneumatically operate the shale discharge valve. This automatic shale extraction has proved very reliable and has been used at three plants in the U.K.

2.4.2 Shaking Tabling

Shaking tables have long been used for cleaning coal. In 1960, in the U.S. alone, 45 Mt of coal were washed on shaking tables.[107] In most operations the feed is in the 9.5 mm - 0 mm size range. When flotation is also used in the coal preparation plant, the feed is usually deslimed at around 150 μm.[108] In this range, shaking table capacities are up to 12t/h. Even for a finer than 3 mm feed, capacities up to 6t/h may be obtained. Separation is effective down to 0.1 mm. Hydraulic classification may double throughputs. With special riffling, where every fifth or sixth riffle is taller, it is possible to clean coal as coarse as 20 mm.

The operating mechanism of a shaking table utilizes flowing film concentration which is modified by the deck riffling. In flowing film concentration, the lower density coarser and rounder particles are washed downstream by the liquid faster than the denser, finer and more angular particles. Between the riffles, particles at high solids concentrations stratify according to density. Stratified layers, starting from the deck surface, are fine dense, coarse dense, fine low density and coarse low density particles. The appearance of coarse particles in upper layers is due to the sheer created within the layers by liquid flow and by deck motion. The forces which drive coarse particles to upper layers are named after Bagnold who quantified these forces.[109] As the lower density coarse particles appear in the top layers they are first washed over the riffles by the water flow. The sequence of separation on a partially riffled deck is shown in Figure 23. However, on a fully riffled deck the dense mineral of various sizes separates in the opposite sequence.[97] While on a partially riffled deck the fine, dense particles travel the shortest distance across the deck, on a fully riffled deck where the riffles do not taper, the coarser dense particles travel the least distance across the table. This is because when low density particles come to the top first they are still acted upon by the thin film flowing over the riffles and therefore are carried over. The coarse dense particles travel forward and receive support from the riffles along the full length of the deck. As the bed is gradually depleted of low density particles, turbulence and interstitial water velocities are able to wash the dense fines over the riffles. As a result, the dense minerals are spread into bands as shown in Figure 24. The differentiation of middlings and dense minerals is much greater here than on a partially riffled deck. A fully riffled deck has much higher capacity but it does not derive as much benefit from a classified feed as does a partially riffled deck. Particles which tend to report in the same deck zone are difficult to separate. In raw coal washing this applies to fine coal and coarse shale. A hydraulic classifier product is an ideal feed for a shaking table as the classifier would have sorted these particles into the different products. The greater the density of the classifier medium, that is the more pronounced the hindered settling conditions, the easier will be the separation of the classified products on a shaking table.

Shaking tabling can not only remove shale from coal but is also very efficient for the removal of pyrite. Hydraulic classification of the feed prior to tabling is also very advantageous for the removal of pyritic sulphur. When considering intensive coal cleaning by grinding and shaking tabling, closed circuit grinding with a classifier not only reduces fines production but also increases pyrite liberation and gives an easily separable ground product. However, with raw coals containing easily disintegrating shale, a two stage size reduction with tabling, to reject as much shale

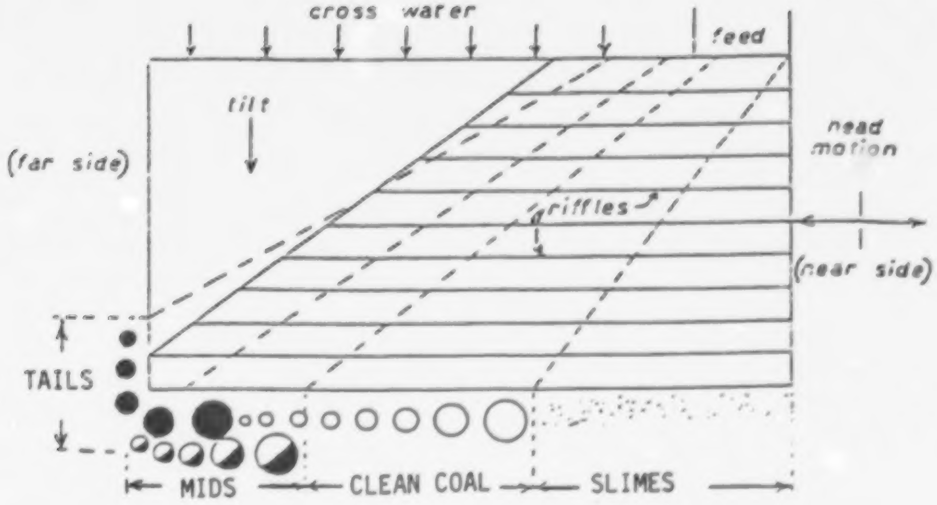

cross water
feed
tilt
(far side)
head motion
riffles
(near side)
TAILS
MIDS
CLEAN COAL
SLIMES

FIGURE 23: The separation pattern of particles on a partially riffled deck.

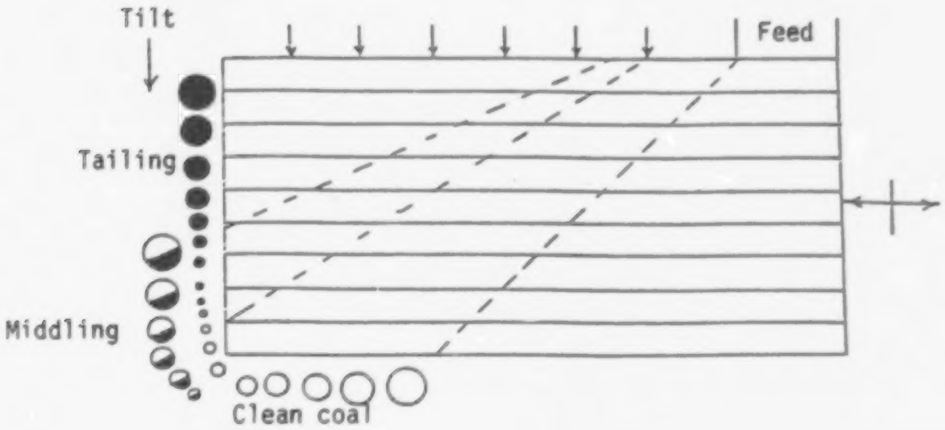

Tilt
Feed
Tailing
Middling
Clean coal

FIGURE 24: The separation pattern of particles on a fully riffled deck.

as possible after the first grinding, followed by further grinding and tabling of the coal product may be necessary. In this form of treatment, a fully riffled deck should not be used as it would place the fine shale with the fine coal.

The operating conditions required for shaking tabling raw coal are well documented.[110] A steady, uniform feed of raw coal and water is essential. The water to solids ratio is usually between 2:1 and 2.5:1 and about 90% of the water reports to the clean coal side of the table. The speed of reciprocation is around 280 strokes per minute with stroke length about 19 mm. The deck surface and riffles are usually made from rubber or polyurethane. The probable error of separation of a shaking table operating with 9.5 mm-0.075 mm coal is about 0.075 g/cm^3 and even when separating 0.03 mm - 0.015 mm fines the error can be as low as 0.16 g/cm^3. The main advantages of shaking tables are:

1. Good separating performance may be achieved under a wide variety of feed conditions. The products are always visible which simplifies adjustment.

2. Great selectivity in pyritic sulphur rejection.

3. Relatively low water to solids ratio.

4. No reagents required.

5. Lower operating and maintenance costs.

To reduce floor area requirements and to simplify feed and discharge arrangements, installations of three or four decks, one above the other, are quite common.[111] A novel approach is to use three twin shaking tables in a triangular arrangement.[112, 113] These units may be stacked into a tower as shown in Figure 25b. A shaking table tower in Germany was recently commissioned for the treatment of finer than 3 mm coal. The tower consists of twelve planes each of six tables driven by six planes of three drive units. The whole installation is constructed in three similar levels each of which contains twenty four shaking tables driven by six drive units. On the top of the shaking table tower, hydrocyclones remove the finer than 125 μm material and the cyclone underflow product is distributed by rotating slurry distributors to the shaking tables. The clean coal is dewatered by centrifuging. The feed rate to each table is 3.2t/h. Shaking tables in similar configurations are used in other West German coal preparation plants. The additional liberation from crushing the coal to less than 3 mm and then treating on shaking tables makes it possible to prepare a clean coal with 43.9% yield at 6% ash, a middling product of 21.7% yield with 43.7% ash and a tailing with 79.3% ash. The clean coal and middlings product sell for 35% more than the untreated fines feed.

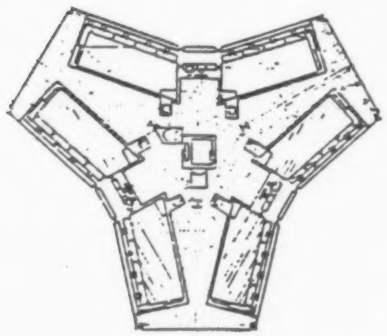


FIGURE 25a: Layout of twin shaking tables in triangular arrangement.

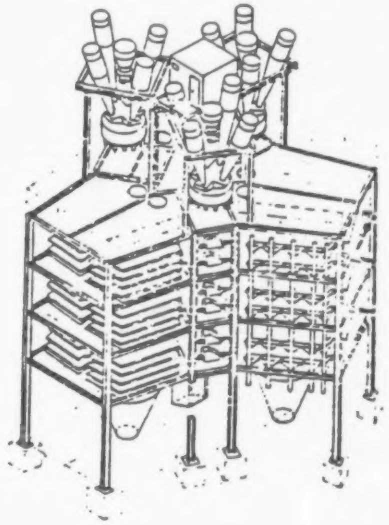


FIGURE 25b: General view of shaking table tower.

Wider use of shaking tables in Eastern Canadian coal preparation plants should be considered, especially for the removal of pyritic sulphur. Shaking tables are capable of good selectivity in both ash and pyritic sulphur reduction of coal fines. They should give better fine pyrite removal than can be achieved by using dense medium processes. The installation of shaking tables in a compact tower form, integrated with desliming cyclones and slurry distributors, appears to be advantageous as this arrangement results in space savings and reduces vibrations transmitted to the building. The tower design makes inspection of deck operation less convenient and homogenization of the feed is most advisable to avoid the necessity of frequent resetting of decks.

2.4.3 Spirals

Even in the 1940's, the industry was interested in recovering coal from refuse and the first application of spiral concentration of coal was in this field.[114] Spirals with flatter pitch are more suitable for coal cleaning. Humphreys Engineering Company produces Model 24C-6, which is a six turn spiral with flatter pitch suitable for coal. The coal slurry of 20%-35% solids by weight is fed to the top feed box. As the slurry flows downwards on the spiral, the lower density particles experience lower frictional forces and travel faster and the centrifugal force moves them radially outwards, while denser minerals spiral down on the inner radii. On the Humphreys spiral some of the dense component is removed through ports in the spiral. In its earliest form the spiral was cast from steel in one third turn spiral elements. Later the elements were made from fibre glass with rubber or polyurethane lining. Two start spirals double throughputs for the same floor area and have been used in Australia for twenty five years.[115] The fibre glass construction reduces the weight of a twin spiral to about 70 kg. More recently other spirals specifically designed for fine coal cleaning have been developed by

Mineral Deposits Limited, Australia. They are the Reichert Marks 9 and 10 spirals, obtainable in Canada through Sala Machine Works Limited, Mississauga. These one piece spirals are made from fibre glass and spray coated with polyurethane for good wear resistance. The trough profiles are designed for selectivity of separation and for elimination of washwater. The Mark 9 spiral is capable of operating over a wide range of feed refuse contents.[116] The trough has a continuously curved profile, with varying cross section, (Figure 26). The Mark 10 spiral is designed for coals of low to medium refuse content and for finishing duties. The point of intersection of the inner and outer sections of the trough profile moves radially outwards from the top to the bottom of the spiral, (Figure 27). Both of these coal spirals have a diameter of 696 mm. larger than mineral spirals (590 mm), with a pitch of 273 mm rather than 387 mm. Their overall height is 2145mm.[117] The clean coal yield of these models may be set by adjustable splitters at the base of the spiral. Integrated units of up to 14 double spirals and containing all the necessary pulp distributors are available.

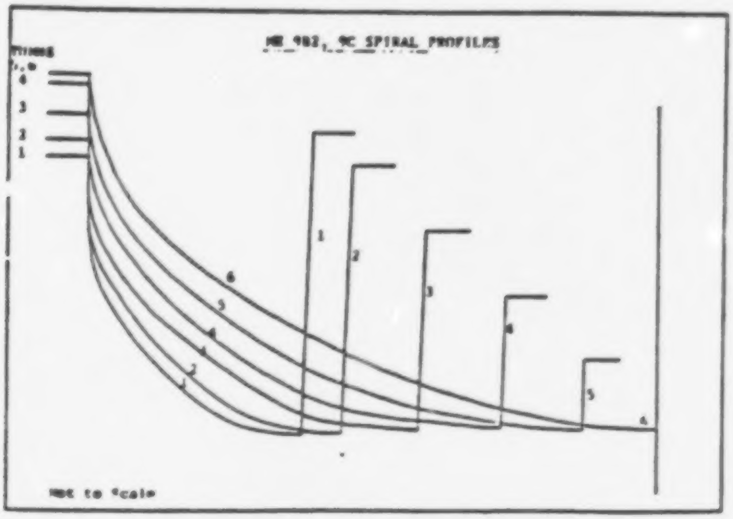


FIGURE 26: Trough profiles for the Reichert Mark 9 Spiral. Turns are numbered from top of spiral.

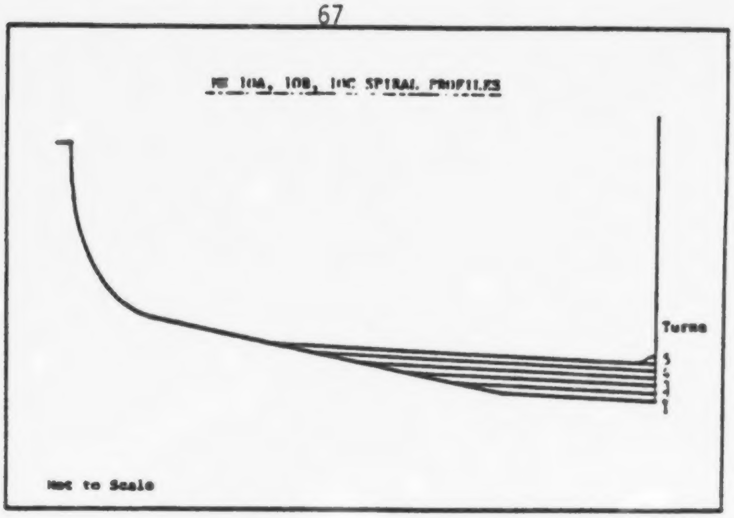


FIGURE 27: Trough profiles for the Reichert Mark 10 Spiral. Turns are numbered from top of spiral.

Like all other gravity concentrators, spirals operate best on prepared feeds within an optimum size range. Spirals will accept feed from 3 mm downwards and are effective in separating down to 75 μm. However, depending on the washability characteristics of the coal, separations above 1 mm may not be satisfactory. The spirals will treat 2t/h solids for every start. As separation is affected by the feed-rate, constant, uniform feed is essential. The feed solids concentration is usually between 20% - 40% solids by weight. Thus they use more water than shaking tables. The Reichert spirals have been field tested in Australia, South Africa and the U.S.A.. The probable error of separation for these spirals varies between 0.09 and 0.11 g/cm^3. They achieve substantial pyritic sulphur removal.[118] A Mark 10 spiral operated on flotation feed material of 20.6% ash 1% sulphur at 2.2t/h reduced the ash content to 5.7% and sulphur to 0.71% at a clean coal yield of 90.9%.[116]

A Mark 9 spiral operating on flotation tailing of 32.5% ash and 5.4% sulphur at 2.7t/h reduced the ash content to 11% and sulphur to 0.89% at a yield of 70.6%.

The main areas of application envisaged for coal spirals are:

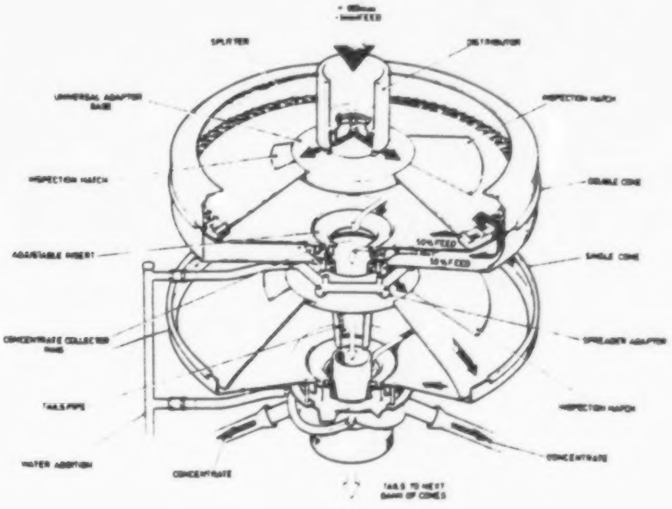
1. Substitution for fine dense medium cyclones.
2. Substitution for coarse flotation.
3. Substitution for water only cyclones.
4. Scavenging rejects from waste streams containing appropriately sized fine coal.
5. Retreatment of fine coal dumps.

Spirals offer the following advantages over other coal cleaning methods:

1. Simple construction and low maintenance requirements.
2. Low capital costs.
3. No reagent or medium required.
4. Low operating and labour requirements.

2.4.4 Reichert Cones

The Reichert cone concentrator was developed about twenty years ago by Mineral Deposits Limited of Queensland, Australia, for the separation of rutile, zircon and ilmenite from beach sands. A typical cone assembly consists of 2 m diameter 17° cones stacked in a frame up to 7 m high, (Figure 28). The separator cones may be single or double. Double cones are generally used in preconcentration and single cones in cleaning. The mineral slurry of 55% - 70% solids concentration is delivered to the top distributor cone at 60 - 90t/h.[115] The distributor cone spreads out the feed on the outer circumference of the separator cone. The slurry flows radially inwards down the cone and is stratified with the denser minerals occurring in the lower layers. The thickness of the slurry layer increases towards the centre of the cone. The dense, stratified material is removed through an annular orifice near the centre of the cone. Cone assembly can be made up in several configurations to achieve the required roughing, cleaning or scavenging operation. The simplest arrangement is a double cone followed by a single cone equipped with an adjustable insert to vary the product yield. In this simple configuration the dense mineral is concentrated on the double cone and cleaned on the single cone. A more complex configuration may include three double cones and five single cones with adjustable inserts, which may be arranged in various sequences,[119] (Figure 29). Recently a new model of a Reichert cone with 3.5 m diameter and a solids capacity of up to 300t/h has been introduced.


SPLITTER
DISTRIBUTOR
UNIVERSAL ADAPTOR BASE
INSPECTION HATCH
INSPECTION HATCH
DOUBLE CONE
ADJUSTABLE INSERT
SINGLE CONE
CONCENTRATE COLLECTOR RING
SPREADER ADAPTOR
TAILS PIPE
INSPECTION HATCH
WATER ADDITION
CONCENTRATE
CONCENTRATE
TAILS TO NEXT BANK OF CONES

DSV CONE CONCENTRATOR STAGE

FIGURE 28: Exploded view of the Reichert Cone Concentrator.

Other uses of Reichert cones in the minerals industry include preconcentration of cassiterite and gold, iron ore beneficiation and concentration of titano-magnetite.[120] Reichert cones do not give sharp separation with a single pass, but repeated passes improve separation sharpness. They are high capacity devices with low capital costs, low

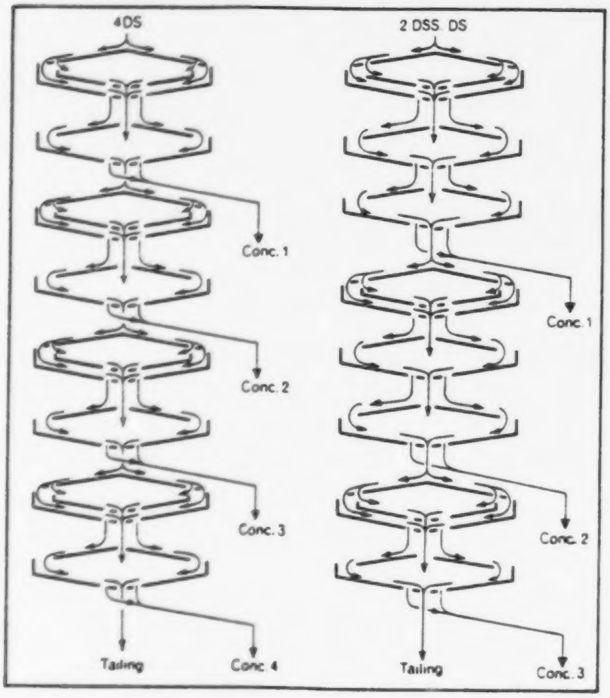


FIGURE 29: Two different flow configurations of a Reichert Cone Concentrator.

floor area requirements and minimal supervision and maintenance needs.

Reichert cones have not been used industrially for coal cleaning although some tests may have been carried out. When used in coal treatment their capacities are likely to be less than half those reported for minerals and they will have to be used with lower solids concentrations by weight. Due to their high capacities and low capital and operating costs these devices should be adaptable for coal cleaning. Cones are expected to give a reduction of pyritic sulphur content in conjunction with ash reduction. Due to the smaller density differences in coal slurry constituents the 3.5 m model, which gives longer separation time, may give better separation results. The testing of Reichert cones on actual fine samples of Eastern Canadian coals should be carried out to assess their suitability.

The testing of full scale Reichert concentrators requires a large quantity of feed material as the cones must be tested at their optimum feed rate. Mineral Deposits Limited have developed a smaller test facility which uses pinch sluices instead of cones, (Figure 30). The Reichert Mark VI Tray Test Assembly will accept 3-5t/h solids and may be used for the prediction of separation performance of full sized cones. Although the tray tester was developed as a test facility, some models have been installed in operating plants.

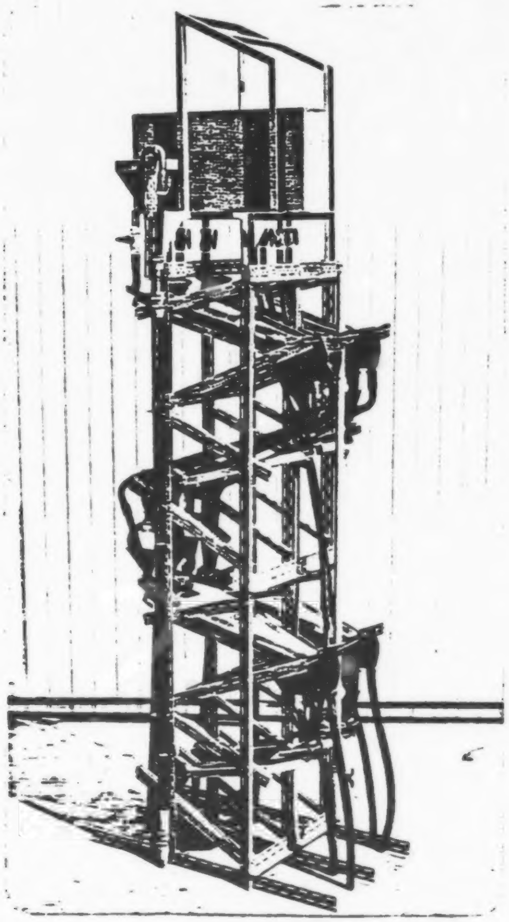


FIGURE 30: The Reichert Mark VI Tray Test Assembly.

2.4.5 Pneumatic Concentrators

In pneumatic separation air is the separation medium and therefore these processes do not need subsequent solid/liquid separation, which is a great advantage. However, the feed to a pneumatic separator must have low moisture content, otherwise capillary adhesion of particles will prevent effective separation. When compared with wet processes, pneumatic separation has low settling ratios due to the low density of air. (See Figure 17c, page 40). But the low viscosity of air also results in comparatively high sedimentation velocities and faster separation. Thus, it follows that pneumatic separators should be operated on narrow feed size ranges of about 2.5:1 and with higher frequencies of oscillation, between 300-600 per minute.

There are several types of pneumatic machines which have been used for fine coal cleaning. As the machines are adequately discussed by Symonds and Butcher, they will be only briefly mentioned here.[121] The riffled air tables are similar in construction and operation to wet shaking tables, but their decks are covered with a fine mesh screen cloth through which flows a constant velocity upward air current. The longitudional oscillations and the upward flow of air stratify the layers. Lower density particles in the upper layers move faster across the deck and pass into the clean coal discharge. Refuse particles travel further to discharge near the end of the deck.

The counter-flow separator uses a rectangular, unriffled deck inclined at about 10° to the horizontal. The deck is vibrated to give a vertical component of velocity which transports the particles nearest the deck upwards. The air flow and vibration stratify clean coal in upper layers and refuse in the lower. Thus, separation takes place as the refuse is transported to the upper end of the deck and the clean coal moves down to the lower end. The counter flow separator may also be considered as an air jig with a movable screen.

Air jigs operate with pulsating air currents and their operating principles are very similar to those of wet jig washers. Air is pulsated through a plenum chamber by rotating flatter valves. The stratification of the bed occurs as a result of air pulsation and oscillations imparted to the deck. In this type of separator the refuse is removed in three stages and the stratified bed is then split into clean coal and middlings product.

The need for closely sized feed for air tables and air jigs may be overcome by increasing the density of the separation medium by introducing fine dense medium material into the bed. With sufficient dense medium, the process becomes a float and sink separation. Sand has been used for this purpose, in which case, the fine sand is screened out from the products. Magnetite medium may be recovered by low intensity, dry magnetic separation. In these processes, the surface moisture of feed particles must be very low to avoid high medium losses. In dry dense medium processes, the sharpness of separation of fine particles is usually low and this greatly limits the usefulness of the method.

A pneumatic gravity separation device, the counter-current fluidized cascade, (CFC), has been developed jointly by the Faculty of Engineering Science, University of Western Ontario, and the Coal Mining Research Centre, Edmonton.[122, 123,124] The device uses a fluidizing medium of limestone or sand mixed with hematite or magnetite to dry-clean coal in sizes down to 0.6 mm. Coal sized between 0.6 mm and 0.1 mm may be cleaned without added medium. The CFC operates on the principle of the physical phenomenon that a vertical particle segregation occurs in a fluidized bed of mixed granular solids when the fluidizing velocity is only moderately in excess of the minimum fluidizing velocity. In the CFC, verticle particle segregation is enhanced in the horizontal direction by use of the counter-current enrichment principle. The counter-current enrichment is accomplished by creating opposite horizontal motions of the upper and lower strata of the fluidized bed by means of a baffled chain that travels through the lower strata towards the jetsam end, (Figure 31). This induces a counter flow of the upper strata solids towards the flotsam end of the CFC. Although there is some difference in bed composition in the vertical direction, this is not as great as the composition changes horizontally. The pilot scale model of the counter-current fluidized cascade is 190 mm wide, 609 mm deep and 5487 mm long. The air distribution grid consists of perforated steel sheets. The raw coal, with the medium when necessary, is fed through a central tube and the air is withdrawn from the CFC to a cyclone and bag filters. The device is maintained under slight vacuum to prevent dust escape. A vibratory trough at each end controls the product withdrawal. The baffles of the transport chain are 185 mm wide, 60 mm high and spaced at 102 mm intervals. The paddles travel at a speed around 0.025m/s. The counter flow is aided by inclining the CFC a few degrees by raising the jetsam end. The medium is recovered from the product by screening. The counter-current fluidized cascade has given quite promising results in tests. Separation of 20 mm - 6 mm raw coal feed with fluidizing medium resulted in a separation probable error of only 0.035 g/cm^3 at a separation density of 1.54. The probable error of separation for a 3mm - 1 mm feed at a cut point of 1.83 g/cm^3 without fluidizing medium was 0.21 g/cm^3. Capacities up to 40t/h per metre width have been achieved. Useful beneficiation also occurs with particles in the 0.6 mm - 0.1 mm size range. The moisture content of the feed should be kept below 2% Compared with wet coal cleaning, CFC is advantageous because it gives a dry product with very little size degradation. Industrial trials are necessary to evaluate the potential of this promising dry separation method.

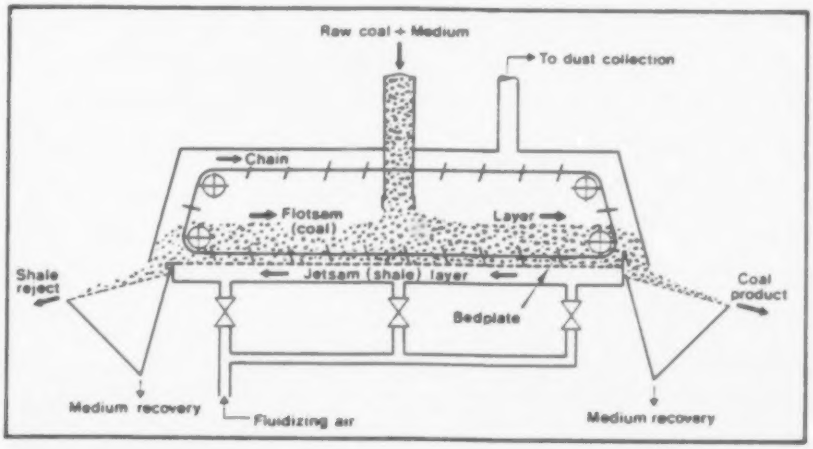

Raw coal + Medium
To dust collection
Chain
Flotsam (coal)
Layer
Shale reject
Jetsam (shale) layer
Coal product
Bedplate
Medium recovery
Fluidizing air
Medium recovery

FIGURE 31: The counter-current fluidized cascade.

2.5 Dense Medium Separation

In dense medium separation particles with lower density than the medium float while those of higher density sink. Although disposable dense media have been used, magnetic media which are regenerated by magnetic separation are most commonly employed. In coal preparation magnetite medium is adequate but for the higher separation densities required in mineral processing ferrosilicon is also used

The efficient removal of dense medium solids from the products and their subsequent regeneration from the dilute medium is essential for the reduction of medium losses and for efficient separation. Dense medium particles should be sufficiently fine to form an adequately stable suspension. The sedimentation velocity of a suspension also depends on its solids concentration. Thus lower solids concentrations used at lower medium densities require fine medium solids. In centrifugal dense medium separators, greater forces exist than in static separators and, therefore, they require finer medium solids, between 65% - 90% finer than 45 μm, to prevent segregation.[125] In practice the loss of finer medium particles is greater than for coarse particles.[126] In mineral processing 75% of the total physical loss of medium can be attributed to magnetic separators. In coal cleaning one would expect greater medium losses, due to adherence of medium particles to the porous coal, than occur with adherence to minerals. Since the fine particles constitute most of the losses, which are not balanced by degradation of the remainder, the dense medium can become too coarse for efficient separation. This applies particularly to dense medium cyclone operation. Thus the dense medium circuit always contains coarser medium than the fresh make-up medium. It is commonly found that during plant start-up and initial operation magnetite consumption is exceptionally high, reaching 10 kg/t. This is partly explained by the retention of medium in every corner and crevice of the dense medium circuit and also because the new magnetite contains weakly magnetic and nonmagnetic materials which are not recovered by magnetic separation. However, the fines content of the new magnetite medium must also contribute to this increased loss. Medium solids often contain some titano-magnetite, ilmeno-magnetite, or hematite inclusions. These lower magnetic susceptibility particles may account for 10% - 20% of the medium solids. Wet magnetic laboratory tests should be used to determine this proportion, which may be the apparent cause of excessive medium losses. The fines content of the prepared magnetite medium feed could be reduced by grinding in closed circuit instead of in batch grinding.

The Australian Coal Industry Research Laboratories Ltd., (ACIRL) have conducted a series of tests to identify the properties of magnetite which govern their losses in dense medium separators in coal preparation plants.[126,127] Equipment suppliers usually claim that magnetite losses should be around 0.5 to 1.0 kg/tonne of raw coal. However magnetite losses exceeding 5 kg/tonne have been reported. The reasons for these excessive losses are often not clearly defined and may be due to variations in the quality of magnetite supplied. At ACIRL the magnetite was extensively tested, both in the laboratory and plants. Laboratory tests included determination of particle size, density, apparent magnetic susceptibility, coercive force and saturation moment. Microscopic and chemical analyses and Davis tube tests were also carried out. The contributory factors in magnetite losses were considered to be:

1. Breakdown of the magnetite-hematite structure when such composite crystals occur, the free hematite being unrecovered by the magnetic separator.

2. Presence of non-magnetic material in the feed.

3. Excessive quantities of minus 10 μm magnetite in the original supply.

4. Incorrect feed conditions for the magnetic drum separator.

5. Insufficient wash water on the rinse screen.

6. Inadequate draining and rinsing screen capacities.

A simple mathematical model was developed based on the consideration of forces experienced by a magnetite particle while passing through a magnetic drum separator. The model indicates that the important properties of magnetite which affect losses for the sizes ranging between 30 and 8 μm are particle size, density, apparent magnetic susceptibility and porosity, particle size being the most important.

In coal preparation plants the medium density is often controlled by differential pressure devices or dip tubes. While these devices may give adequate density control, in mineral processing magnetic and radioactive measuring methods are also used and are regarded as more accurate and reliable.[128] The magnetic method uses Ramsay coils around the medium carrying pipe. The magnetic medium passing through the coil generates a current indicative of the quantity of magnetic material present. This method is not suitable for density control of mixtures of magnetite and ferrosilicon. The radioactive density control utilizes the attenuation of radiation by the medium solids. A radioactive source is placed on one side of a medium carrying pipe and a detector on the other side. The reduction of radiation intensity is a measure of solids concentration within the pipe and is used for density control. The usefulness of these methods should be evaluated for coal cleaning. Accurate density control is more important with increasing amounts of near separation density material in the feed.

Efficient washing of products during their passage over the washing screen is essential in the reduction of medium losses. In practice, the washing sprays are often blocked by solids particles present in the wash water which increases medium losses. The hydrocyclone filter unit, Figure 19, mentioned in section 2.3, might be of value in removing solids particles from the spray water.

The viscosity of dense medium is an important variable influencing separation sharpness. In plant practice, the dense medium viscosity is rarely measured or used for control purposes in spite of its great significance. As the viscosity of medium increases, the separation velocities of particles diminish. Near separation density particles tend to move very slowly, thereby reducing separation efficiency. The viscosity of a suspension increases with increasing solids concentration and increasing fineness of suspended solids. Although a small amount of clay slimes is useful for stabilisation of the medium, an excessive accumulation is detrimental and must be removed. Every 10% of fine slimes present in the medium reduces the sedimentation velocity of separating solids by more than half.[129] A further disadvantage of high slime content is that it makes media markedly non-Newtonian with a definite yield strength. Due to the shear strength of the medium, there is a limiting force below which there will be no particle movement. The shear strength results in an anomalous viscosity whose value depends on shear rate. Fast moving particles experience lower viscosity of medium than slow moving ones.

The properties of dense media may be controlled by the separate or combined use of reagents and vibration.[130] The addition of reagents disperses the suspension, reduces structure formation and consequently lessens the shear strength of the dense medium. Vibrating the medium at 3 to 5 cycles/s with 4 to 6 mm amplitude has been shown to speed up the motion of particles whose densities are near the separation density and thus substantially increases separator capacity and improves separation efficiency. Application of vibration to dense medium may allow the use of a lower density, cheaper medium, e.g. magnetite instead of ferrosilicon. Industrial tests revealed that in the dense medium concentration of a lead ore the tailings lead content was reduced and the separation efficiency increased when vibration was applied.

In lower density magnetic suspensions, 3% - 6% of clay by weight is often necessary to improve medium stability by reducing sedimentation velocities of medium particles. But the clay content increases the yield strength of the medium, which is undesirable. The use of dissolved polymer instead of clay has been investigated for the stabilisation of dense media.[131] It was found that polymer Kelzan XC stabilized the dense medium at less than 0.1% dosages without unduly affecting the viscosity and shear strength of the suspension. Media of lower density, below 1.75 g/cm^3, needed more polymer for stabilisation. Polymer addition would permit the use of coarse medium particles with lower medium losses. Another advantage of polymer use is said to be ease of start up as the media will not settle out rock hard.

The rheological properties of mineral slurries in gold and uranium extraction have been studied.[132] In this work the viscosity was measured by means of a Brookfield LVT viscometer and empirical equations were found to express the effects of solids concentration, shear rate and temperature on the relative viscosity of the slurry. Although this approach is useful for predicting the rheological properties of a particular slurry, composition and size variations will change the constants in the equations. It should not be difficult to take direct measurements of rheological parameters either in situ or on a sample of slurry. In the future this should become commonplace both in coal preparation and mineral processing as the importance of rheological properties in any wet separation process cannot be overemphasized.

The evaluation of dense medium separation performance is usually carried out by float and sink analysis of products. From the data obtained, the Tromp distribution is determined and the separation density (cut point) and probable error of separation are evaluated. Float and sink analysis is a time consuming operation and therefore performance evaluation is rarely carried out. De Beers Diamond Research Laboratory, Johannesburg, South Africa, developed a time saving method for performance evaluation of dense medium separators through the use of "density tracers".[133,128] The tracers consist of small pieces of plastic material, colour coded according to densities. To evaluate the performance of dense medium separation, the tracers are introduced with the feed. After separation, the plastic tracers are recovered from each product, separately sorted by colour and counted or weighed. The Tromp partition coefficient may be calculated as the ratio of the number of pieces of a certain colour appearing in one product to the total number of pieces of that colour. Thus, the separation density and probable error may be determined. This method is reported to reduce the duration of the whole test to half an hour. Tracers are invaluable in investigating the effects of certain operating variables on separation efficiency. They make it possible to keep a close check on plant operation. However, tracers do not give all the information as provided by true analysis and therefore float and sink analysis should also be retained for more thorough testing. The plastic from which the tracers are made was developed and patented by Diamond Research Laboratory. A heavy filler is added to a conventional plastic in very carefully controlled proportions to obtain the required densities. Tracers are available from Debex of Johannesburg. Density tracers should be used for the rapid evaluation of dense medium coal cleaning results.

If routinely used, the technique would assist in the maintenance of peak performance.

2.5.1 Static Dense Medium Separators

The separating vessel of a dense medium separator may be a cone, a tank or a rotating drum. The various vessel shapes, together with the different methods of product removal, give a large number of possible configurations. The various static dense medium separators and their operation are well documented.[134] A properly run dense medium bath will give high sharpness of separation, with probable errors between 0.020 - 0.035 g/cm^3. The separation efficiency is lower at excessive throughputs and when the medium viscosity is allowed to increase through the build up of fines. Since static dense medium separation is an important method of coal cleaning, this area has been well investigated and little additional information can be supplied from mineral processing practice.

2.5.2 Dense Medium Cyclones

Dense medium cyclones are commonly used for cleaning coal in the 25 mm - 0.5 mm size range. A 600 mm diameter dense medium cyclone, operating at 5m - 6m head, will treat 80t/h of raw coal.[135] The probable error of separation is usually between 0.02 and 0.04 g/cm^3.[125]

A study to assess the factors affecting the efficiency of separation in dense medium cyclones was carried out to identify important operating parameters and to define criteria for prediction of separation response.[136] The test work used DSM cyclones and Vorsyl separators with magnetite-ferrosilicon mixtures as dense media. The apparent viscosity of the media was shown to be affected by the size, density, particle shape and volumetric concentration of the medium solids as well as by the shear rate. The settling properties of the medium governed its segragation in the cyclone. It was found that the medium segregation did not seriously affect separation performance as long as the density difference between underflow and overflow media was below 0.5 when the density of feed medium was around 2.8 g/cm^3. The separation density increased with decreasing size of the feed. The increase in separation density was proportional to the inverse diameter to a power of 1.3. The tests also confirmed that the separation density of coarse particles was very close to the density of underflow medium. The above findings demonstrate that dense medium cyclones cannot be regarded as perfect dense medium devices which separate solely according to density irrespective of particle size. While large particles may separate at 2.8 g/cm^3, small particles of 0.2 mm may separate at 3.8 g/cm^3. Although the above work was carried out for higher medium densities than used in coal preparation, the general principles should also apply to lower medium densities. Thus in dense medium cyclones finer raw coal particles separate at higher densities and consequently at higher ash contents. This emphasises the need for narrow size range feeds for improved separation sharpness. The medium segregation effects in centrifugal dense medium separators should be investigated more fully since segregation significantly affects the performance of these separators.

Although dense medium cyclones are capable of separating coal down to 75 µm particle size, normally sizes down to 0.5 mm - 0.6 mm only are treated, to permit easy separation of products from medium. Test performance of a 203 mm diameter dense medium cyclone on finer than 3.33 mm raw coal demonstrated this, as the probable error of separation was 0.1 g/cm^3, while for the coarse end the probable error decreased to 0.024 g/cm^3.[137,138] When dense medium separation is used for washing down to zero, the dense medium solids must be separated by magnetic separation. In these cases, the magnetite medium should be ground finer for better separation of coal fines and magnetite losses may be 2 kg/t or more.[139]

Washing raw coal to zero with dense medium cyclones is established technology in the U.S.A. where at least three plants are operating on this principle.[140, 141] In these plants the fines products (usually below 0.6 mm) pass through the drainage and rinsing screens and are fed separately to magnetic separators for magnetite recovery. The fine refuse is thickened and discarded while the fine clean coal slurry is sized at 150 μm over sieve bends and the coarser fraction is dewatered in centrifuges. The finer than 150 μm fraction may be froth floated and filtered, or directly filtered and blended into the coal, or discarded. The three flow diagrams of figure 32 illustrate the variations of this scheme. Figure 32a shows a 90t/h circuit for 10 mm - 0 feed. Figures 32b and c are for the treatment of 38 mm - 0 feed at 180t/h. These circuits are said to be simpler and less costly than conventional dense medium plants treating the same size ranges.[140] However, these schemes have the following disadvantages:

1. The medium specific gravity is not instantly adjustable downwards over a wide range.

2. The separation specific gravity cannot be independently controlled for coarse and fine coals.

3. Magnetite losses in some of the plants are more than in conventional dense media cyclone circuits.

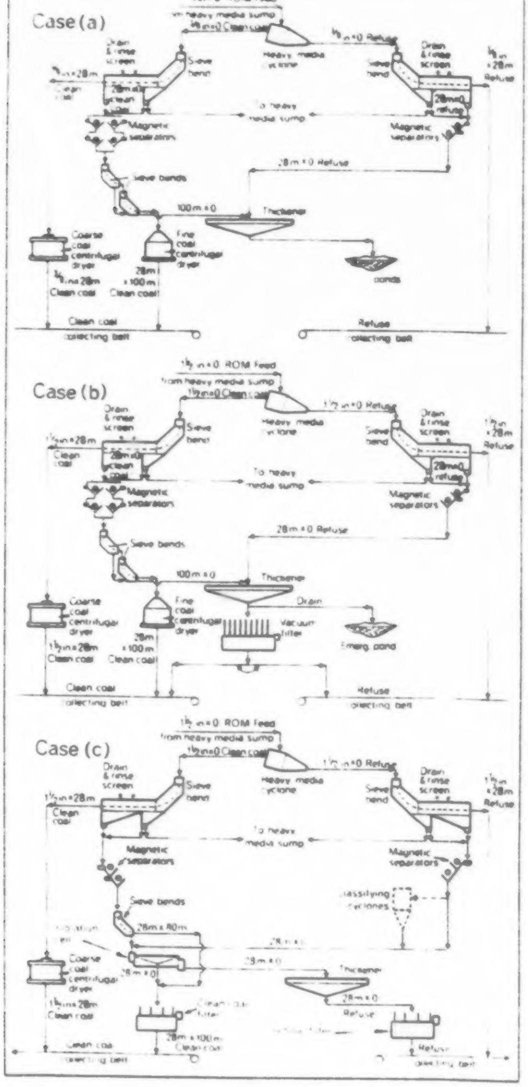

Case (a)
½ in × 0 ROM Feed
from heavy media sump
½ in × 0 Clean coal
½ in × 0 Refuse
Drain & rinse screen
Sieve bend
Heavy media cyclone
½ in × 28m Clean coal
28m × 0 clean coal
½ in × 28m Refuse
28m × 0 refuse
To heavy media sump
Magnetic separators
28m × 0 Refuse
Sieve bends
100m × 0
Thickener
Coarse coal centrifugal dryer
Fine coal centrifugal dryer
28m × 100m Clean coal
Ponds
Clean coal collecting belt
Refuse collecting belt
Case (b)
1½ in × 0 ROM Feed
from heavy media sump
1½ in × 0 Clean coal
1½ in × 0 Refuse
Drain & rinse screen
Sieve bend
Heavy media cyclone
1½ in × 28m Clean coal
28m × 0 clean coal
1½ in × 28m Refuse
28m × 0 refuse
To heavy media sump
Magnetic separators
28m × 0 Refuse
Sieve bends
100m × 0
Thickener
Drain
Vacuum filter
Emerg. pond
Coarse coal centrifugal dryer
Fine coal centrifugal dryer
28m × 100m Clean coal
Clean coal collecting belt
Refuse collecting belt
Case (c)
1½ in × 0 ROM Feed
from heavy media sump
1½ in × 0 Clean coal
1½ in × 0 Refuse
Drain & rinse screen
Sieve bend
Heavy media cyclone
1½ in × 28m Clean coal
1½ in × 28m Refuse
To heavy media sump
Magnetic separators
Classifying cyclones
Sieve bends
28m × 80m
Flotation cell
28m × 0
Thickener
Coarse coal centrifugal dryer
Clean coal filter
28m × 100m Clean coal
Refuse filter
Refuse
Clean coal collecting belt
Refuse collecting belt

FIGURE 32: Washing raw coal to zero with dense medium cyclones

The Greenside coal preparation plant in South Africa has built an additional section to handle 35t/h of finer than 0.5 mm coal.[142] The plant uses 150 mm dense medium cyclones to process clean coal with 7% ash content.

A new type of dense medium cyclone, the Swirl Cyclone developed by Osaka Shipbuilding Co. Ltd., Tagawa Machinery Works, Japan, is gaining acceptance in industry.[143] Although the Swirl Cyclone is based on the same operating principles as a conventional dense medium cyclone, it is novel in that it has an air vent tube for controlling the vortex (Figure 33). The feed solids and dense medium are introduced to the cylindrical feed chamber through a nozzle with an adjustable opening. The sink product is discharged through the apex aperture into the sink chamber and then through the outlet. The float product subsides through the vortex finder into the float chamber. The air core of a conventional cyclone is frequently unstable; here it is stabilised by means of an air vent open to the atmosphere. The height of the air vent is variable as is that of the vortex finder. Swirl Cyclones are available in sizes from 100 mm to 750 mm diameter. The largest size is capable of treating 90t/h coal feed with up to 75 mm top size. Tests carried out with a 350 mm diameter model on Japanese bituminous coal with a very high ash content (41.5%) and a high percentage of near separation density material show that the performance of the Swirl Cyclone is equivalent to those of standard dense medium cyclones. When treating raw coal feed 32 mm to 0.5 mm at a separation specific gravity of 1.36, with a feed rate of 11.8t/h the probable errors of separation for 32-12 mm, 12-6 mm and 6-0.5 mm fractions were 0.023, 0.017 and 0.022 respectively. On increasing the feed rate to 32.8t/h and maintaining the same separation specific gravity the probable errors for the above size ranges were 0.020, 0.022 and 0.020. In the above tests a coarse magnetite dense medium was used with 15.7% plus 100 mesh and 20.5% minus 325 mesh material. Tests carried out with fine magnetite medium, (93.2% minus 325 mesh) made no difference to cyclone performance. The ability of the Swirl Cyclone to operate with coarse medium is said to reduce magnetite losses as they mainly occur in the finer size ranges. Tests carried out at a separation specific gravity of 1.73 gave separation efficiencies similar to those obtained at 1.36.

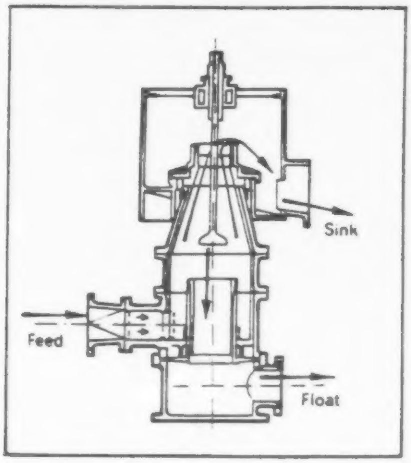


FIGURE 33: Cross-section of the Swirl Cyclone

Three-product dense medium cyclones were developed in the U.S.S.R.[144] These cyclones are constructed from wear resistant material or have carborundum linings and are used in coal preparation plants. Their applications are reported to be in fine coal washing, jig middlings treatment and the recovery of very clean coal. The three-product dense medium cyclones avoid the need for two product cyclones in series, dispense with the dense medium circuit and simplify plant layout and operation.

Dense medium cyclones are also useful for pyritic sulphur removal. The Stinson plant of the Beaver Creek Consolidated Coal Company, U.S.A., treats the raw coal in three size ranges.[145] The 38 mm to 9.5 mm coarse coal is treated in a dense medium vessel at 1.32 g/cm^3 separation density. The 9.5 mm to 0.6 mm fraction is treated in dense medium cyclones at around 300t/h at 1.3 g/cm^3 density. The finer than 0.6 mm portion is treated by shaking tables and flotation cells. The total sulphur content of the feed is around 3% with about 0.6% organic sulphur. The dense medium vessel reduces this to 0.69% S, the dense medium cyclone to 0.71% S and the shaking tables to 0.98% S. The sulphur content of the froth floated clean coal is 0.78%.

2.5.3 Vorsyl Separator

The Vorsyl separator was developed by the Mining Research and Development Establishment of the National Coal Board, U.K. and has been used in coal preparation plants since 1967. The Vorsyl separator is a cylindrical centrifugal device of 600 mm diameter, incorporating a tangential feed inlet, a vortex finder and an annular opening called throat, (Figure 34). The raw feed and the dense medium enter through the tangential inlet and the clean coal leaves through the vortex finder. The denser material passing through the throat enters the shale chamber and discharges into the vortextractor through a tangential inlet. The pressure energy is dissipated in the vortextractor and the centrifugal forces exerted on the medium supply backpressure which prevents emptying the separator. The throat opening must be large enough for the coarsest sink material to pass. The volumetric flowrate through the throat is independent of the throat size but is adjustable by changing the vortextractor outlet. A Vorsyl separator is capable of washing 600t/h of 13 mm - 0 raw coal at a probable error of separation of about 0.04 g/cm^3.[135] The separation performance of a Vorsyl separator is similar to that of a Dutch State Mines (DSM) dense medium cyclone or to that of a Dynawhirlpool separator. In 1980, the Mining Research and Development Establishment scaled up the Vorsyl separator to 720 mm diameter.[68,69] The larger unit is also cast from Nyhard but has an involute feed inlet. The scaled up unit will treat 100t/h finer than 30 mm raw coal. Tests conducted at Maltby colliery indicate that the separation performance is at least as good as that of the smaller unit. Although the scaled up unit is capable of treating 50 mm maximum feed size the probable error of separation of the 2 mm - 0.5 mm fraction was found to be only 0.043 g/cm^3 at 1.4 g/cm^3 separation density in the Maltby tests.

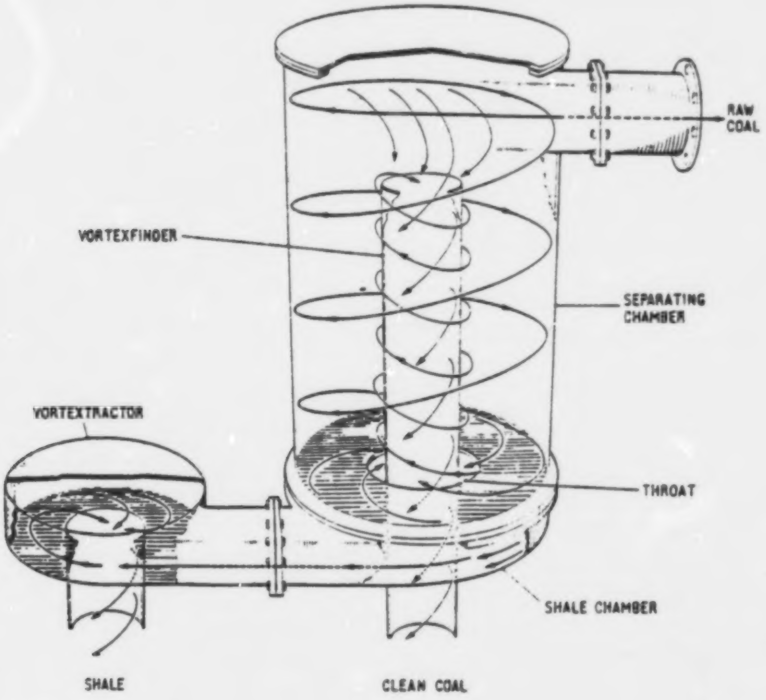

RAW COAL
VORTEXFINDER
SEPARATING CHAMBER
VORTEXTRACTOR
THROAT
SHALE CHAMBER
SHALE
CLEAN COAL

FIGURE 34: Schematic diagram of the Vorsyl separator.

2.5.4 Dynawhirlpool Separator

The Dynawhirlpool (DWP) separator, manufactured by Minerals Separation Corporation, was developed in the early 1960s for the concentration of metalliferous ores.[146, 147] Its use has been extended to beneficiation of coal and industrial minerals, particularly in the United States, South Africa and Australia. The separator consists of a tube equipped with an axial inlet and outlet and also a tangential inlet and outlet and is usually operated at an inclination of 15°

to 45° to the horizontal, (Figure 35). The feed enters at the top, through the axial inlet together with a little medium while about 90% of the dense medium is pumped through the lower tangential inlet. The lower density product is recovered through the axial outlet and the higher density product through the upper tangential outlet. Separators are made with diameters up to 460 mm and handle up to 100t/h of 50 mm to 0.5 mm feed.[148] In South Africa at Landau the DWP separator is used in multi-stage operation. The 6 mm to 0.5 mm coal is fed at 150 t/h to three 400 mm dia. DWP units in parallel which reject 30% of the feed at a specific gravity of 1.55. The remaining 70% is split into two products of roughly equal quantities, in three separators which form the secondary stage, separating at a specific gravity of 1.35 to produce a steam coal middlings of 13% ash and a low ash coal of about 7% ash.

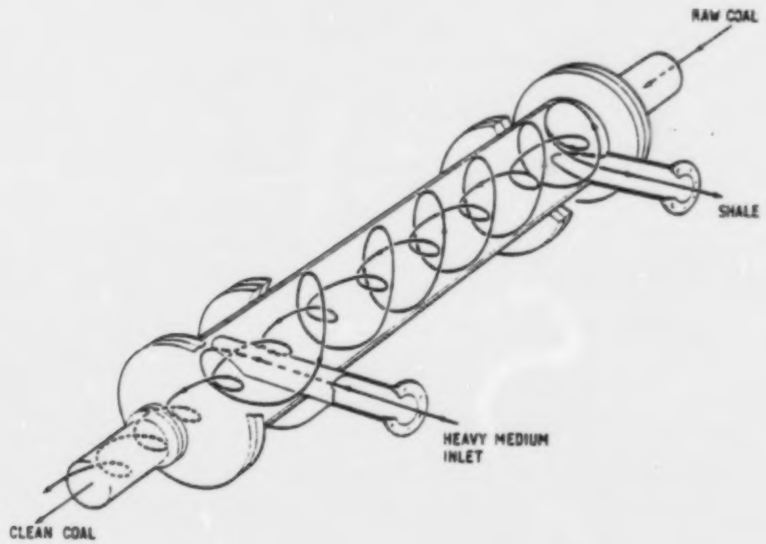


FIGURE 35: Schematic diagram of the Dynawhirlpool separator.

2.5.5 Tri-Flo Separator

The Tri-Flo separator, which is effectively a two stage Dynawhirlpool separator with some improvements, was developed in 1977. The prototype of the 350 mm diameter Tri-Flo separator was installed in the Masau plant, Sardinia, as a replacement for the standard Dynawhirlpool separators used in the preconcentration of Pb-Zn sulphide ore.[149] The Tri-Flo separator has been installed in at least nine plants for the separation of lead/zinc, fluorspar, iron, fluorspar/galena, fluorspar/barites, wolframite/cassiterite, bauxite and coal. The machine has two involute medium inlets, two involute sink product outlets, an axial feed inlet and an axial float product outlet, (Figure 36). The tangential inlets and outlets have been improved to give less turbulence. The feed enters the inclined separator vortex at atmospheric pressure. The separator may be operated with a single dense medium, producing a float and two sink products with slightly different separation densities.[150, 151, 152] When the Tri-Flo separator is operated with two dense media of differing densities, the sink products are separated at individually controllable densities within a single vessel. The two stage treatment using a single dense medium improves separation in one of two ways. Either the two sink products may be combined to increase recovery of dense minerals or the removal of the second sink product from the floats increases the grade of the low density mineral. The first case is a scavenger action for dense minerals while the second is cleaning of low density minerals. The two stages also give sharper separation with a lower probable error. Short term variations in feed characteristics are usually detrimental in dense medium separation as the automatic control is not fast enough to correct the effects of feed variation. In a two-stage system the feed variations are less harmful because the second stage treats the product of the first stage and therefore operates under more constant conditions. The second sink product may be recrushed and after desliming may be returned for retreatment. This scheme reduces fines production and improves recoveries. The first installation with two separately controlled media densities is at the Prestavel plant in Northern Italy, treating plus 1 mm fluorspar ore containing galena. Tri-Flo separators are versatile and appear to have advantages over single stage dense medium separators. They are available in four vessel sizes from 250 mm to 500 mm with capacities of 15t/h to 90t/h. The floor area for the 90t/h complete installation is 9 m x 16 m with a maximum height of 12.5 m.

A 100t/h Tri-Flo separator with a 500 mm diameter has been installed for coal washing in Salzburg Austria. This unit uses a single density medium and produces three products. The applicability of Tri-Flo separators for dense medium separation of Eastern Canadian coals should be considered. Using a single separator with dense media of two different densities simplifies plant layout, while using Tri-Flo with a single dense medium would give improved separation efficiency, or an additional middlings product.

2.5.6 Stripa Process

In 1953, Svensson demonstrated that density separation may also be made in a high solids concentration bed of sand sized particles in motion.[153] In a bed appreciably higher densities may be achieved than in a suspension. The separation process which uses an autogenous medium is known as the Stripa process and has been used for the concentration of iron ore in Sweden, Angola, China and Malaysia. There is also a Stripa plant in Turkey concentrating chromite and one in Norway which has treated pyritic copper ore for 15 years. The operation of the Stripa process is shown

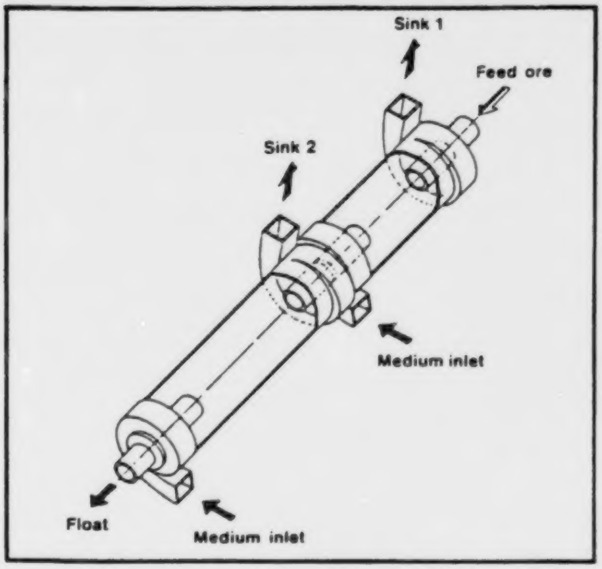


FIGURE 36: Schematic diagram of the Tri-Flo separator.

schematically in Figure 37. The separation takes place in a shaking trough (4). The feed is introduced from a cone (13) at about 50% solids by volume. The cone serves as a thickener and surge tank for the medium which is obtained from the ore fines. In the trough the sand size medium settles rapidly to form a thick bed (7), while the rest of the medium floats above the bed as a suspension (8). The feed plate (3) distributes the ore evenly across the width of the trough. Dense material sinks to the bottom of the trough and is transported at about 0.1 m/s while the float overflows at about 0.5 m/s. The bottom of the tank is perforated and divided into several sections along its length, each with its own separately controlled water supply. Most of the water is added at the feed end and there is gradually less and less introduced towards the discharge end. When the water supply is increased, the bed flows more rapidly, reducing its thickness. The water also washes the finer particles from the bed into the suspension layer, reducing the viscosity of the bed so that separation occurs more rapidly. At the discharge end of the trough, the separating plate (9) divides the bed into float and sink. The products are drained and rinsed on a washing screen (10) using the cone overflow water. The quantity of medium circulating is about 3 to 5 times the volume of crude ore although pumping power is below 1 kW/t of feed. The medium is not cleaned since gangue particles do not unduly interfere with separation. These particles collect mainly in the upper layers of the bed due to water flow. About 10 kg of medium/t of ore treated is removed to the fines treatment plant and replaced by fines extracted from the concentrate.

The largest Stripa unit built has a trough 1.25 m wide and 6 m long with a capacity of 100 to 150t/h on a 60 mm to 6 mm feed. The length of stroke of the trough is 20 mm at 360 strokes/min. Swedish plants operate on iron ore at Stripa, Grängesberg and Strassa at 3.2, 3.68 and 3.07 separation densities respectively. The probable error of separation of the Stripa process for iron ore is around 0.1 g/cm^3.

The probable error of separation for coal is expected to be lower as in coal washing the bed densities and solids concentrations are considerably lower. According to Svennson, the simple, cheap and trouble-free operation of the process more than compensates for the lower separation efficiency when compared with conventional dense medium separation. The Stripa process is a high capacity separation method, of relatively low cost, which needs no magnetite medium. The shallow separation bed, together with the vibration which tends to reduce medium viscosity, contribute to the high throughput of the Stripa process. This process may be of use in coal washing and in reclaiming clean coal from refuse tips.

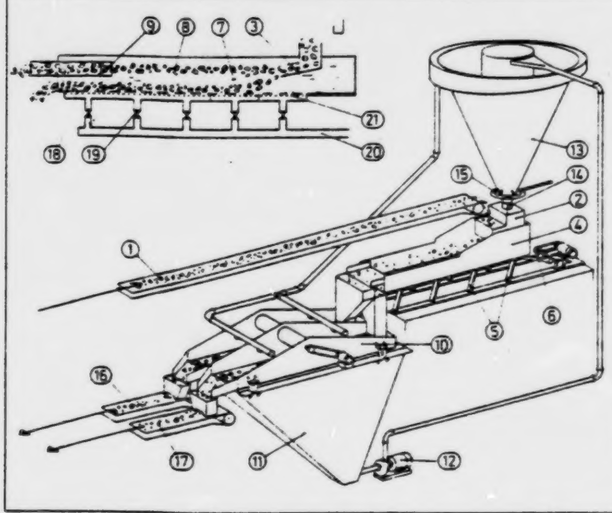


Fig. 6. Stripa plant:
1 - Ore conveyor;
2 - Feed box; 3 - Feed plate; 4 - Trough;
5 - Spring rods;
6 - Tie-rods from eccentric; 7 - Bed;
8 - Suspension layer;
9 - Separating plate;
10 - Washing screen;
11 - Pump sump;
12 - Medium pump;
13 - Cone thickener;
14 - Valve;
15 - Dilution water;
16 - Conveyor for floats; 17 - Conveyor for sinks; 18 - Bottom plates; 19 - Valves;
20 - Water pipes;
21 - Water box.

FIGURE 37: Schematic diagram showing operation of Stripa process.

2.5.7 Otisca Process

The Otisca process, developed in 1975, by Otisca Industries Limited, is a dense medium process which uses organic liquids as the separation medium. The main advantages of using organic liquids instead of aqueous suspensions are:

1. The organic liquid of low boiling point will evaporate leaving the products dry.

2. The sharpness of separation is extended to fine particles and fines are separated at similar densities to coarse particles.

3. The lower viscosity of organic liquids compared to that of magnetite medium facilitates fast and accurate separation.

The ideal organic separating liquid should have the required density, low viscosity, low boiling point, should not react with coal, should be non-toxid, non polar and relatively cheap.[154] Freon 11 and Freon 113 satisfy most of the above requirements. The separation performance of the Otisca process has been tested in laboratory and in pilot scale. Laboratory test results showed that separation of 75 µm to 0 raw coal in organic liquids gave a clean coal yield versus specific gravity relationship which was very close to the theoretical washability figures.[155] A simplified flow diagram of the Otisca process is shown in Figure 38. This circuit was used in an 18t/h pilot plant at the North Branch Mine of the Island Creek Coal Company, West Virginia. After separation the products are dried and the organic liquid is regenerated. In 1979, the capital investment cost was assessed as U.S. $1100/tonne per hour and the operating costs as below U.S. $1.10 tonne of feed processed.[156]

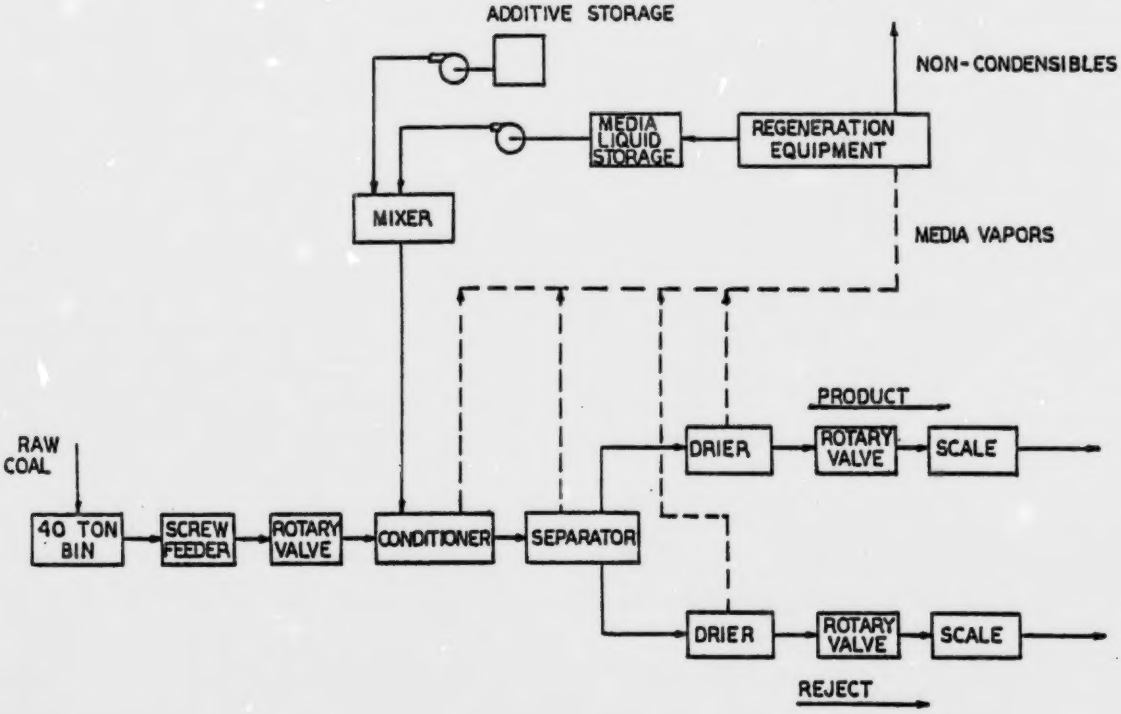

ADDITIVE STORAGE
NON-CONDENSIBLES
MEDIA LIQUID STORAGE
REGENERATION EQUIPMENT
MIXER
MEDIA VAPORS
PRODUCT
RAW COAL
DRIER
ROTARY VALVE
SCALE
40 TON BIN
SCREW FEEDER
ROTARY VALVE
CONDITIONER
SEPARATOR
DRIER
ROTARY VALVE
SCALE
REJECT

FIGURE 38: A simplified flow diagram of the Otisca process.

A 113t/h plant was built at the Muskingham mine in Beverly, Ohio.[157] This has been operating at full capacity and has exceeded the planned separation performance. The process is said to be simple, versatile and inexpensive to operate and uses a non-toxic medium. Significant reductions of ash and sulphur contents have been obtained. The Otisca process shows promise of being an improved dense medium process with several advantages over dense medium separation methods using suspensions. The operating costs will determine whether this process will be widely accepted.

2.6 Water-Only Cyclone

An ordinary classifier hydrocyclone can give a certain amount of separation, according to density, on a narrow feed size range. Since, in a classifier hydrocyclone, separation takes place in substantially free settling conditions, the settling ratio is low and the process is not true density separation. When the configuration of a hydrocyclone is modified to have a short wide-angled cone and a relatively long vortex finder, the cyclone will accumulate high and intermediate density solids. Then, due to the high solids concentration and the resulting higher settling ratios, the cyclone will operate more like a dense medium separator. This type of cyclone is known as a water-only or autogenous dense medium cyclone. Water-only cyclones were specially developed for treating coal and are summarised here only for completeness.

Even with autogenous media, water-only cyclones do not give very sharp separation, as the probable error is reported to be from 0.12 - 0.28 g/cm^3.[158] Another disadvantage is the limited feed size range, 6 mm - 0.15 mm, or for better separation 1.0 mm - 0.15 mm, they are able to treat. Below 150 μm their cleaning efficiency falls off. Their separation density cannot be regulated easily as the accumulation of autogenous media is a function of the cyclone design. Thus feed size and density distributions significantly affect separation.[159] Water-only cyclones tend to separate at or above a density of 1.8 g/cm^3. With improved design the separation density may be reduced.

Liller developed a cyclone which is said to separate 50 mm to zero coal according to specific gravity in the range of 1.2 to 1.7 SG cut points.[160] In this cyclone, the feed inlet is designed to have a critically deflected flow through a narrow inlet restriction which facilitates proper circulation within the cyclone, (Figure 39). The vortex finder, which has an internal sleeve, is long to prevent short circuiting of the middlings. For best operation, the proper choice of design and operating variables to suit the feed characteristics is essential. The main factors in design are feed inlet restriction, deflection angle, bottom geometry and vortex finder depth. A 457 mm diameter Liller cyclone, operating with water medium, is claimed to be able to treat 14-122t/h raw coal. A 254 mm diameter model is rated at 5-41t/h. Liller cyclones may be made from alumina, polyurethane or Nihard. The alumina cyclone has a life of 200,000 t washed coal, polyurethane 40,000 t, and Nihard 60,000 t. The cyclone apex is replaceable although the alumina apex lasts as long as the body, but the polyurethane needs replacement after 3,000 t, and the Nihard apex after 1,500 t. Replacement takes only five minutes. If the claims for the Liller hydrocyclones are substantiated in long-term industrial operation, they represent a break through which should change the future of coal treatment.

Water-only cyclones are useful for the removal of pyritic sulphur and for cleaning oxidised coals. They have a shortcoming in that their separation efficiency decreases rapidly towards the finer sizes. The smallest particle size effectively cleaned is about 150 μm.[161] Desliming of the clean coal product is important because most of the feed water accompanies the clean coal, taking with it most of the very fine material. Thus the fines have to be treated separately, possibly by flotation. For successful operation the percentage of rejects should be less than 40% of the feed and the refuse particles should not be flat or slabby. A flow diagram of a typical water-only cyclone circuit is shown in

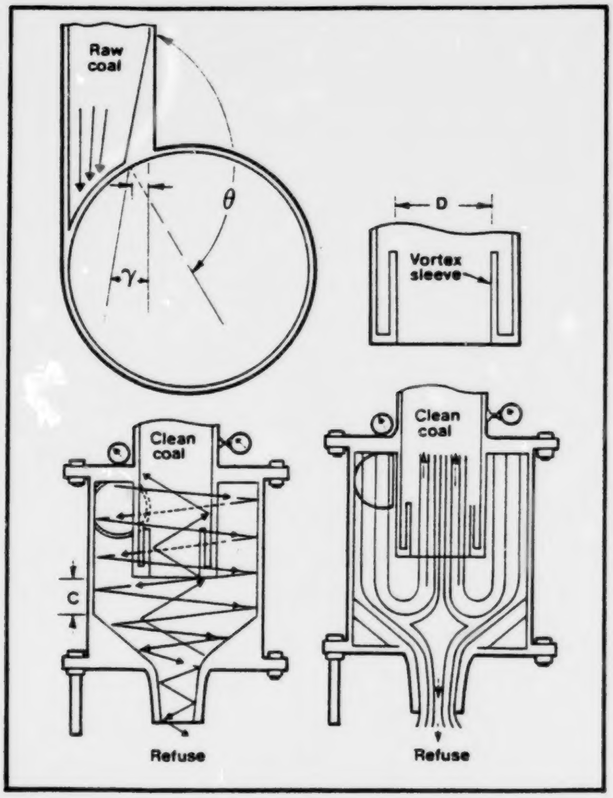


FIGURE 39: Schematic diagram of the Liller Cyclone.

Figure 40. Here the primary cyclone underflow is scavenged by the secondary cyclone, whose overflow is recirculated to the feed of the primary cyclone while the underflow is discarded. The minus 104 μm fraction is removed from the clean coal by classifying hydrocyclones and froth floated in a single bank of flotation cells.

Goodrich shows through a theoretical analysis that this system cannot reach the efficiency and yield of dense medium cyclones but gives higher efficiency than water-only cyclones in conventional circuits.[162] When producing relatively high ash clean coal the system works nearly as well as dense medium cyclones. However, it is unacceptable for the production of low ash clean coal. Since the scalping system produces a lower clean coal yield at a specific ash content than dense medium cyclones the economy of the system must rely on the lower cost of the process. A typical flowsheet, showing water only cyclones scalping the feed to dense medium cyclones is given in Figure 41.

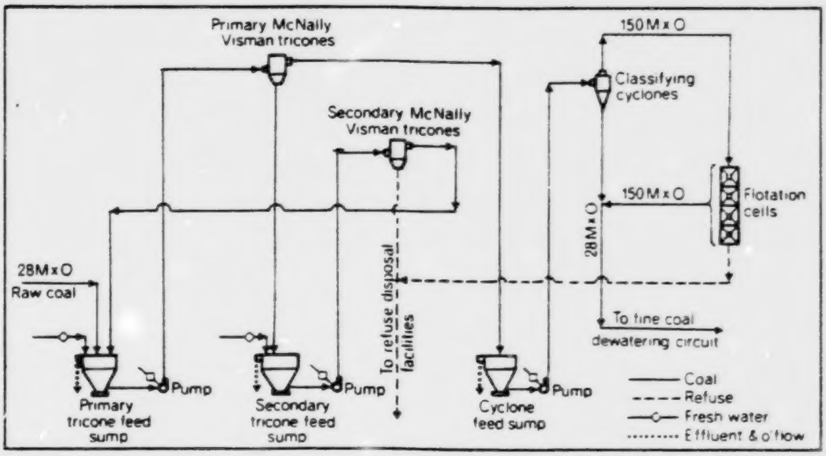

Primary McNally Visman tricones
150M x O
Classifying cyclones
Secondary McNally Visman tricones
150M x O
Flotation cells
28M x O
28M x O
Raw coal
To refuse disposal facilities
To fine coal dewatering circuit
Pump
Primary tricone feed sump
Pump
Secondary tricone feed sump
Pump
Cyclone feed sump
Coal
Refuse
Fresh water
Effluent & o'flow

FIGURE 40: Flow diagram of a two-stage water-only cyclone circuit.

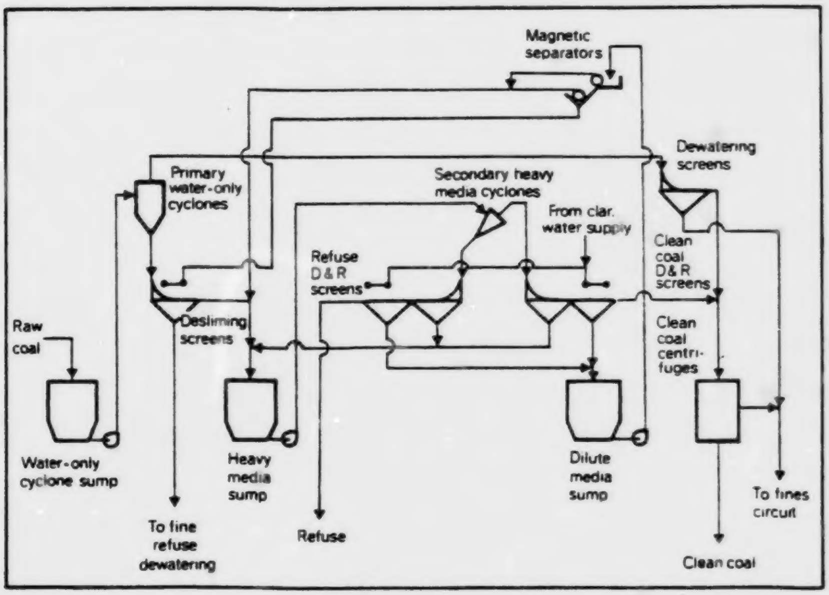

Magnetic separators
Dewatering screens
Primary water-only cyclones
Secondary heavy media cyclones
From clar. water supply
Refuse D&R screens
Clean coal D&R screens
Raw coal
Desliming screens
Clean coal centri-fuges
Water-only cyclone sump
Heavy media sump
Dilute media sump
To fine refuse dewatering
Refuse
To fines circuit
Clean coal

FIGURE 41: Use of water-only cyclones in feed scalping.

2.7 Magnetic Separation

Magnetic separation techniques have so far had little application in coal preparation. Their main use is restricted to regeneration of magnetite from dilute medium circuits, by low intensity wet magnetic separators. The same separators have been used in dense medium washing of coal to zero size. The removal of tramp iron from run-of-mine coal is another use for magnetic separation. Interest in high intensity, high gradient magnetic separation for pyrite sulphur removal is a recent development. Magnetic separation techniques have frequently been reviewed.[163-168] Every substance has some magnetic properties. Ferromagnetic substances, such as iron, magnetite, ilmenite are very strongly affected by magnetic fields. Paramagentic substances possess very low positive susceptibilities and are attracted towards positions of higher magnetic field intensities. Diamagnetic substances have very low negative susceptibilities and are repelled from high magnetic field intensities. The mass magnetic susceptibilities of coal, pyrite and shale in cgs. emu/g are respectively, $-(0.4 \text{ to } 0.6)10^{-6}$, 0.3×10^{-6} and $(39 \text{ to } 45)10^{-6}$.[169]

The magnetic force on a particle is proportional to the product of particle volume, difference between magnetic susceptibilities of particle and medium, the magnetic field intensity and the field gradient. Thus there is no force on a particle in a uniform magnetic field, no matter what the field strength. But high field gradients cannot be produced over large distances and consequently the working volume of high intensity, high gradient separators is restricted. Magnetic separators may be divided into:

1. Low or medium intensity separators, used in wet or dry forms for the separation of strongly or medium strongly magnetic minerals.

2. High intensity, high gradient separators used with weakly magnetic minerals.

3. Super conductive separators intended for weakly magnetic and para or diamagnetic minerals.

4. Magnetohydrostatic separators operate with magnetic fluids and produce float and sink type separation.

2.7.1 Low and Medium Intensity Separators

As magnetic separation is partly responsible for dense medium losses the correct choice of efficient magnetic separators for the regeneration of magnetic dense media is important. Permanent magnet separators are most commonly used for this purpose. In circuits where the recovered magnetite is densified before its return to the dense medium sump, the densifier overflow may be responsible for some of the magnetite loss.[170] The Electro-Permanent Wet Drum Separator, with a 1220 mm diameter drum, is an improved separator designed for magnetite media recovery and is available from Eriez Magnetics, Pennsylvania.[171]

In this design strong magnetic fields exist even several centimetres away from the drum surface. Permanent magnetic materials, capable of generating magnetic field densities and gradients similar to those occurring in induced roll separators are commercially available.[172] These samarium-cobalt permanent magnetics may help to reduce magnetite losses. Due to the higher field densities and gradients generated by the magnets, both the capacity of the separator and magnetite recovery are increased. Some industrial models of separators using samarium-cobalt magnets are available for dry separation.[173] It is expected that wet separators of this type, which could be employed advantageously in dense medium regeneration circuits will soon be available.

The magnetic hydrocyclone thickener was developed by Boxmag-Rapid Ltd., Birmingham, England, after a long period of research.[174, 175] The effect of a magnetic field surrounding a hydrocyclone has been the subject of much research by many investigators, but for various reasons, previous attempts to utilize the effect were not successful. In Boxmag-Rapid's design it was recognized that the apparatus would not be a true magnetic separator because the sizing action of the cyclone on non-magnetic particles would remain unaffected by the magnetic field, but a correctly positioned and shaped magnetic field could enhance the sizing and separation of magnetic particles. The magnetic hydrocyclone thickener is availalbe in two sizes. The MCTU65 is 65 mm in diameter with a throughput of 6 m^3/h at an inlet pressure of 75 kPa (10.9 lb/in^2), while the MCTU200 is 200 mm in diameter and has a throughput of 30 m^3/h at an inlet pressure of 59 kPa (7.2 lb/in^2). The power consumption of the magnets is 150 W and 2.5 kW respectively. Boxmag-Rapid foresees applications for the magnetic hydrocyclone in various areas. In fine iron ore circuits the magnetic hydrocyclone would replace conventional cyclones and thickeners for the recovery and thickening of magnetite at increased unit capacities and negligible mineral loss. Recoveries of 99.5% were reported with a single pass. In dense medium separation plants the use of magnetic hydrocyclone thickeners would reduce the size of wet magnetic drum separators needed, by dewatering the dilute medium from the washing screens before magnetic separation.

In the alternative design of model MCTO, all magnetic material is discharged with the overflow product. This type of magnetic hydrocyclone thickener should have application in iron ore concentration and dense medium plants since this cyclone recovers most of the magnetic material in the overflow product, thus eliminating overgrinding of the magnetic fraction and reducing the grinding load.

2.7.2 High Intensity High Gradient Magnetic Separators

High intensity high gradient magnetic separators have matrix elements present in a magnetic field. The function of the matrix is to produce high intensity convergent magnetic fields in the vicinity of the elements which then attract and capture magnetic particles from the feed flowing through the matrix. Most non-magnetics are unimpeded on their passage through the matrix. The magnetics are usually washed inbetween pole pieces with low pressure water to remove adhering non-magnetics. The magnetics are removed from the rotating matrix ring at the neutral magnetic zone by high pressure water jets. There are several designs of matrix type separators, the earliest of which was the Jones machine. The Boxmag-Rapid separator has a verticle field, while the Krupp-Sol separator has tangential magnetic fields. In the Sol separator the magnetic field is generated within a solenoid and not inbetween magnetic pole pieces as in the other models.[176] Comparison has been made of the separation performances and capital and running costs of these machines.[177] An interesting development is the Eriez high intensity separator, designed for uranium recovery from gold tailings, as this machine uses loose ball matrix, part of which is continuously removed for cleaning.[178] Cleaning is necessary because the feed contains fibrous material which blocks the matrix pores. The rate of ball removal in the neutral magnetic zone is controlled so that the entrapped fibrous material and ferromagnetic particles are kept to an acceptable level. Highly magnetic particles such as magnetite should be removed from the feed to the high intensity separators as they tend to clog the matrix.

High gradient magnetic separation is applicable to fine coal beneficiation as it is capable of removing shale and pyrite into the magnetic product. Pilot scale studies carried out in the U.S.A. show that substantial ash and sulphur reductions may be achieved in this way.[179, 180] Compared with flotation, high gradient magnetic separation gives much higher yields at given ash or sulphur reductions.[181]

Mathieu carried out wet high gradient magnetic separation tests on Devco Prince coal of 9.26% ash and 2.56% pyritic sulphur, using a Sala separator.[182] These tests indicated that for a feed size of 150 µm to zero the clean coal ash may be reduced to 5.2% and the pyritic sulphur content to 1.17% at a yield of 75.7%. This represents 57.4% ash and 65.4% S removal. Both wet and dry magnetic separation were also tested on 300 µm - 600 µm and 150 µm - 300 µm fractions on Prince and Lingan coals.[169]. There was little difference between the results of the wet and dry methods. However, the dry method requires no solid/liquid separation and consequently is less costly. The Lingan coal responded better to high gradient separation for ash reduction because of its greater degree of ash liberation, but it gave slightly lower sulphur reductions. When compared with dense liquid separation at 75% yield, the efficienty of magnetic separation was 84.6% for ash removal and 90.1% for sulphur rejection. Dry high gradient separation of U.S. coals was investigated by Oak Ridge Laboratory, Tennessee.[183, 184] In these tests a Sala high gradient magnetic separator was operated at 0.9t/h on 150 µm - 45 µm feeds. For the Western Kentucky Seam No. 9 coal the ash content was reduced from 13.8% to 9.5% and the sulphur content from 6.5% to 4% at 90% heating value recovery. The Lower Kittaning Lady Jane coal's ash content of 11.7% was reduced to 8% and its 3.5% sulphur to 2.3% at 90% heating value recovery.

2.7.3 Superconducting Magnetic Separators

In high gradient magnetic separation the maximum magnetic induction is limited by the saturation of magnetic poles which occurs at about 2T (20 kG). In superconducting separators the magnetic field is generated without ferromagnetic material by a refrigerated superconductive solenoid and, therefore, higher magnetic field densities may be achieved. With an Nb-Ti superconductive coil, magnetic fields up to 8 T (80 kG) are easily attained.[185,186,187] A superconductive machine is light and compact compared with a high gradient magnetic separator. It requires less power by a factor of 10 to 50. The coils must be maintained constantly at liquid helium temperature of 4.2°K. So far superconductive magnetic separators have hardly been used industrially for mineral processing. When the technology becomes proven this technique should have direct application in coal fines beneficiation for ash and sulphur reductions.

2.7.4 Magnetohydrostatic Separation

Magnetohydrostatic (MHS) separation utilises the forces acting on a magnetic fluid in a static magnetic field gradient. The magnetic fluid may be a manganese, iron or cobalt salt solution which forms a paramagnetic liquid or an ultra fine ferromagnetic suspension of iron, ferrite or magnetite in aqueous or organic liquid, called ferrofluid.[188-191] Commonly used paramagnetic liquids include manganous nitrate, manganous chloride, ferrous chloride, cobalt chloride and ferrous nitrate. The magnetic field gradient exerts forces on the magnetic fluid in the direction of increasing field strength. Particles present in the fluid experience an apparent density which may be several times as high as the liquid density. While the upper density limit for dense liquids is about 4,500 kg/m^3, the apparent density in MHS separation may rise to 15,000 kg/m^3. The density may be simply regulated by varying the magnetic field strength. Paramagnetic liquids are usually cheaper than dense liquids used in float and sink or density gradient work. In a single operation several components of the feed may be separated. MHS separation is suitable for the examination of small samples or even single particles in a laboratory form. Some difficulties encountered in MHS separation are that highly magnetic minerals or minerals with magnetic inclusions tend to migrate towards the poles and also paramagnetic liquids react with certain minerals.

A laboratory attachment enabling the Frantz type isodynamic magnetic separator to be used as an MHS separator is offered by Chas. W. Cook and Sons, Birmingham, U.K.. With this attachment apparent densities approaching 12,000 kg/m^3 may be obtained using manganous nitrate solution as the paramagnetic liquid.[192] The apparatus may be easily calibrated for routine laboratory use as the logarithm of the apparent supplementary density of the paramagnetic liquid varies linearly with the logarithm of the magnetising current.

Magnetohydrostatic separation is unlikely to be applied to large scale industrial treatment of coal, but the technique is useful in liberation analysis of pyritic sulphur.

2.7.5 Magnex Process

The Magnex process is a dry chemical method for the removal of ash and sulphur from coal by magnetic means.[193] The finer than 1 mm coal is treated with iron carbonyl vapours at a temperature of 150°C - 190°C for about half an hour. The iron carbonyl breaks down above 110°C into iron and carbon monoxide. The iron is selectively deposited on ash forming minerals, but not on the coal.[194,195] Pyrite reacts with iron carbonyl to form a pyrrhotite-like material and carbon monoxide. As a result the shale and pyrite become magnetic and may be removed by medium intensity magnetic separation.

Pilot scale tests on several U.S. coals show that while maintaining over 95% heating value recovery, the Magnex process is able to reject 25% - 54% ash and 57% - 90% pyritic sulphur.[196] Sulphur reduction is generally much better than can be achieved by perfect density separation.

2.7.6 Separation with Magnetic Fluids

Magnetic fluids may be used for coal beneficiation in an alternative way to their use in magnetichydrostatic separation. If a magnetic fluid is selectively adsorbed onto raw coal constituents then it may be used for the enhancement of magnetic properties with subsequent magnetic separation of the components.[197] A magnetic fluid is a stabilised suspension of magnetic particles in a carrier fluid. The particles are generally very fine, around 10 nm. The carrier fluid may be water, hydrocarbons or other liquid. Dispersing agents are also added to aid the selective adsorption of magnetic fluids. The selective adsorption is also enhanced by surface conditioning of solids by various reagents. Magnetic fluids with organic carriers tend to wet coal and therefore coal reports as a magnetic product and shale as a non-magnetic product. Alternatively, a magnetic fluid containing aqueous carrier will adhere on shale and pyrite making these waste products report to the magnetics. Magnetic fluids at a dosage of 1 kg/t gave good ash and sulphur reductions. Much more test work is needed before the viability of this process is proven.

2.8 Electrostatic Separation

Electrostatic separation utilizes the electrical properties of mineral particles to effect separation. The main electrical properties exploited are conductivity, contact potential and dielectric constant and accordingly different types of machines exist to utilize each property.[198]

Normally electrical forces are weak and thus this method is more suitable for particles below 1 mm. Dry electrostatic separation methods become less selective for particles below 50 μm where particle adhesion forces are significant. Fraas gives a good summary of electrostatic separation methods.[199]

2.8.1 Conductivity Separation

Electrostatic separation methods which exploit differences in mineral conductivities are the most commonly used in industry. The separation by these machines may be considered as occurring in three distinct stages: the charging of particles, the separation at a grounded (earthed) surface and separation by trajectories.[200] The particles may be charged by:

1. contacting dissimilar particles,
2. ion bombardment,
3. induction.

The latter two are the more important charging mechanisms using high tensions of 15 kV to 40 kV. Charged particles are attracted to the grounded surfaces of drum or plate they contact. But the particle charges gradually drain to the ground at different rates. The more conductive particles lose charges first, semiconductors later and insulators retain their charges longest. Thus a particle will leave the rotating drum at the point at which the electrostatic force cannot balance the centrifugal and gravitational forces exerted on the particle. Therefore conductors leave first, while insulators leave later or may even have to be brushed off the drum. This process results in different trajectories for the different conductivity minerals and so appropriately placed splitters are able to divide the feed into two or three products.

These machines are made in roll, plate and screen types.[201] Mineral Deposits Limited, Queensland, Australia, supplies these machines in integrated units with high voltage supplies, (Figure 42). The roll type separator is offered in two start three roll units. Within each start, the feed flows vertically through the unit from one roll to the next. The system is equipped with reversible chutes and, therefore, a single unit may be operated in any one of the three modes, namely conductor cleaner, non-conductor cleaner or hybrid.

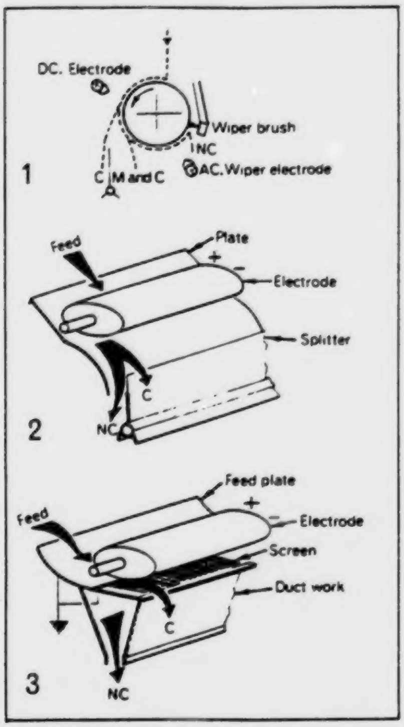

DC. Electrode
Wiper brush
NC
AC. Wiper electrode
1
C
M and C
Feed
Plate
+
−
Electrode
Splitter
C
2
NC
Feed plate
Feed
+
−
Electrode
Screen
Duct work
C
3
NC

FIGURE 42: The roll (1), plate (2) and screen (3) electrostatic separators.

The standard capacity of the unit is 4-6 tonne/h and it will handle feed between 600 μm and 63 μm, but particles coarser than 1 mm should first be removed from the feed. The unit is designed to accept feed temperatures up to 150°C

Both the plate and screen electrostatic separators are available as two start five stage units. The non-conductors flow through the machine and the clean conductor product is progressively removed at each stage. Each module has a standard capacity of between 3-4 tonne/h. The feed requirements are similar to those of roll machines except that feed temperature should not exceed 80°C. The three types of machines complement each other as the products from the high tension separators suit the separation characteristics of the plate and screen separators. High tension roll separators tend to leave fine conductor particles in the non-conductor product. Screen electrostatic separators remove these fine conductors very efficiently. Similarly, the coarse non-conductors are rejected from the conductor product by the plate electrostatic separators. The manufacturers claim that multi stage roll, plate and screen units permit treatment of up to 30 tonne/h feed material in a space less than 5.5 x 3.0 m and 2.5 m high. This space would also accommodate the complete high voltage supply. The design also permits all operation and maintenance functions to be carried out at the front and rear of the module and side access is unnecessary.

In separation by mineral conductivity surface moisture on solids is undesirable as a conductive moisture film prevents separation. High air humidity may cause precipitation of moisture film on particles. To some extent this may be overcome by warming the feed material before separation. Another method used in mineral processing is to make particle surfaces hydrophobic by adsorbing reagents similar to flotation collectors. On a hydrophobic surface moisture cannot form a continuous film and therefore will not greatly affect the overall conductivity of mineral particles.

2.8.2 Contact Potential Separators

When two particles of dissimilar compositions are in physical contact, electrical charges are transferred between them and consequently one particle becomes more positive while the other acquires a negative charge. This charging process is known as triboelectrification. The magnitude of charge transfer depends on the work function of the particles, their conductivities, and the duration of contact. After triboelectrification the charged particles are allowed to fall within an electrical field created between two high tension electrodes. The electrical field deflects the particles during their descent and thus the feed may be split into the required number of products.

Contact potential has been used in laboratory scale to separate quartz and dolomite mixtures.[202] The triboelectrification process took place in a 100 mm diameter nickel plated mild steel cyclone. Due to the friction on the cyclone wall, quartz particles acquired a negative charge while dolomite particles became positively charged. Separation took place between two plates, one earthed and one at minus 30 kV electropotential.

Contact potential has been utilized in laboratory tests for the separation of fine coal from shale.[203,204] Here the triboelectrification of the finer than 150 μm raw coal was carried out in a fluidized bed. When a gas is flowing in upward direction through a dense particulate bed, the fluidized particles mix and collide with each other and with the container walls. This causes charging of particles in a relatively short time. Material for container walls should be chosen to enhance charge separation. The conductivities and surface conditions of particles are important factors in triboelectrification. High temperature and high relative humidity tend to increase surface conductivity and reduce the final charge. Surface oxidation and selective adsorption of some gases or chemicals may improve selectivity of separation by modifying the relative charging of different minerals. Tests were moderately successful reducing ash content from 48% to 30% at 70% clean coal yield and 77% heating value recovery.

The above technique was used in the form of an electrostatic separation tower, on Lingan, N.S. coal, to test whether separation of petrographic components and pyrite rejection were possible.[205,206] Although some separation of vitrinite, fusinite, exinite and micrinite was observed, their differentiation was not good. However, there was a substantial reduction of pyritic sulphur as pyrite tended to be transported towards the positive electrode.

The use of contact potential for ash and sulphur reductions of coal fines is only at the development stage. Many variables affect separation performance and its applicability only lies below 300 μm.

2.8.3 Dielectric Separation

The theory of separation methods based on the use of differing dielectric constants of minerals has been developed by Andres.[207] He derives equations for forces acting on particles in dielectric liquids under the influence of a convergent electric field. Andres showed experimentally that the predicted electrical forces for non-conductive minerals agree well with the measured values but for conductors, which drastically distort the electric field, the measured forces were only about one third of those predicted.

A high tension electrical analogy of the matrix type magnetic separator, the dielectric separator, has been developed and tested.[208] If the magnetic field of a wet high intensity magnetic separator was replaced by an electric field and the magnetic matrix elements were changed to dielectric elements it would be possible to separate minerals of various dielectric constants. The matrix elements of a dielectric separator could be glass spheres of relative dielectric constant of $\varepsilon = 4.5$, but copper wires with a dielectric constant of infinity would induce slightly greater electrical forces. The dielectric constant of the separating liquid should be between those of the minerals to be separated. A suitable separation liquid may be made up from mixtures of carbon tetrachloride and dichloromethane in the 2.25 to 9 dielectric constant range. The higher dielectric constant mineral will adhere to the matrix while the mineral with the lower dielectric constant will pass through the matrix. When the electric field is removed the higher dielectric constant mineral may be washed off to achieve separation. Tests carried out on a mixture of 74 μm quartz ($\varepsilon = 4$ to 4.5) and ilmenite ($\varepsilon = 33$) in equal volume fractions of a carbon tetrachloride and dichloromethane mixture ($\varepsilon = 5$) using 0.1 mm diameter copper wires for the matrix gave 90% recovery for ilmenite and 14% for quartz at 7 kV/cm field strength. The electrodes in the separator are insulated and therefore conduction is prevented and feed moisture does not cause problems.

The U.S. Bureau of Mines has designed a dielectric separator for the continuous laboratory scale separation of minerals.[209] The separator consists of a rotating drum electrode and a concentric wire screen electrode positioned 3 mm apart. The electrodes are immersed in a dielectric liquid and a high voltage is applied across them to produce a high gradient electric field. Minerals of higher dielectric constant than that of the liquid are attracted to the drum electrode and removed, while lower dielectric constant minerals drop through the screen electrode. In tests with 1,000 V at 450 Hz and 8 rev/min drum electrode speed, a rutile grade of 68% with 93% recovery was obtained. The dielectric liquid was kerosene with 7.3 volume per cent of nitrobenzene. The diameter and length of the drum electrode were 26 mm and 38 mm giving a feed capacity of 3 g/min. Tests were carried out on 28 mineral mixtures, 19 of which were adequately separated.

Separation by dielectric constant is not a promising method for raw coals of low economic value. It requires careful choice of the dielectric liquid medium and the operating costs are expected to be higher than coal prices can justify. The technique may be useful for laboratory test purposes.

2.9 Sorting

Automatic sorting of minerals takes many forms depending on the mineral properties utilized and on the presentation of the mineral particles for the detection of these properties. In general, the operation of every sorter includes the following stages:

1. Individual presentation of particles for examination by a detector. Particles must be in a single layer, sufficiently spaced both in the direction of motion and laterally to allow the detector to evaluate each particle separately. This examination may take place on a conveyor belt or during free fall. To aid individual presentation and to keep throughputs high, the maximum feed particle size ratio is kept to about 2 to 1.

2. Detection. When the mineral is presented in single lines, the particles travelling in each line may be evaluated by a static detector. With single layer presentation either scanning is used or a multi-detector system is employed to build up a pattern of particle properties across the belt. From these

properties the particle grade is evaluated and comparison with a preset value determines whether the particle is accepted or rejected.

3. Ejection. Some early sorters used solenoid activated deflectors to divert the detected particles. Most present day sorters have high speed solenoid controlled air valves which can give a very short duration blast of air to deflect particles in free trajectories.

Water jets have also been used. The ejection system must be fast acting and accurate to permit ejection of individual particles. There is a time delay while the particle travels from the sensor to the ejector which must also be taken into account by the electronic circuitry. The delay must be accurate within a few milliseconds. To save compressed air the lesser component of the ore is usually ejected. The detection system may use:

1. Light source (visible, ultraviolet, laser) and detection may rely on reflectance, transparency, colour or fluorescence. Fluorescence may be a natural characteristic of the mineral or may be induced by adsorbed substances.

2. X-ray source with detection through transparency, X-ray or visible fluorescence.

3. Radioactivity which may be a natural characteristic of the mineral or may be induced by irradiation, (e.g. neutron activation, gamma-ray scatter).

4. Conductivity with either direct resistance measurement or remote detection by eddy current sensing.

The basic principles of automatic sorting have been reviewed by King.[210] He gives numerical evaluation of the performance of sorting installations derived from basic principles.

The first high capacity ore sorter was developed jointly by RTZ Ore Sorters and Goldfields of South Africa. Their Model 13 sorter started operation in 1972 at the Doornfontein mine separating gold bearing reef from waste rock.[211] An improved version of the sorter is the Model 16 which was installed at the Yerakini plant, Greece, in 1976.[212-215]

This sorting machine is intended as a preconcentrating device and is normally incorporated in the crushing and screening circuit of the conventional processing plant. Rock must be presented on the sorter belt as a monolayer in such a way that no two pieces touch. Material comes from a small surge bin via two vibratory feeders which control feed rate, (Figure 43). To enhance the optical properties of rock surfaces, water sprays wash the rock as it passes over the second vibratory feeder. Rock pieces travel down a stellite slide plate reaching a speed of 2 m/s. Then the accelerator roller increases the speed to 4 m/s, the speed of the belt, with a consequent spacing of rock pieces. A stabiliser steadies the rock travelling on the belt prior to scanning.

The light source is a helium neon laser. The laser beam is reflected onto the rock stream by a 20 faceted mirror rotating at 6,000 rpm and thus scans the width of the belt. The light reflected back from the rock particles to an adjacent facet of the mirror is transmitted to a photomultiplier tube and converted to electronic signals. With the above mirror speed the scan rate is 2,000/s and with the belt travelling at 4 m/s it is scanned at 2 mm intervals. The electronic signals are transmitted to a processor, which compares the reflection data with pre-set values to determine whether or not the rock should be ejected from the stream. Other data computed includes the sizes of rock pieces and their positions on the belt. This is necessary in order to separate discrete particles from the stream. The particles are ejected in trajectory by means of 40 or 80 compressed air jets positioned across the 800 mm wide rock stream. The system can be set to eject either valuable or waste material. Normally the smaller fraction is blasted to reduce compressed air consumption. The tonnage treated varies between about 15 tonne/h on 10-20 mm material requiring complex recognition criteria to above 180 tonne/h on 80-175 mm feed requiring simple analysis, assuming 2.7 specific gravity of feed. The total cost of photometric sorting calculated for a plant employing five sorters with 7,000 tonne/day throughput was about US$0.60/tonne in 1977.

Model 16 machines have also been used at Mount Carbine Wolfram mine, Queensland, Australia, to separate barren grey schist from quartz containing wolframite and scheelite.[216] Other Model 16 Sorters are used for quartz in Montana, wollastonite in Finland and gold in South Africa at Buffelsfontein and West Driefontein.[217]

Another optical sorter which uses a series of photosensors instead of laser scanning was developed by Gunson's Sortex (Mineral and Automation) Limited, U.K.. The Sortex MP80 machine is probably the first sorter to employ microprocessor technology.[218] The use of high speed electronic components enables this sorter to discriminate very rapidly between different mineral types with good resolution. The sorter can handle 10 mm to 150 mm particle size but a maximum to minimum feed size ratio of 2 is recommended. At the upper

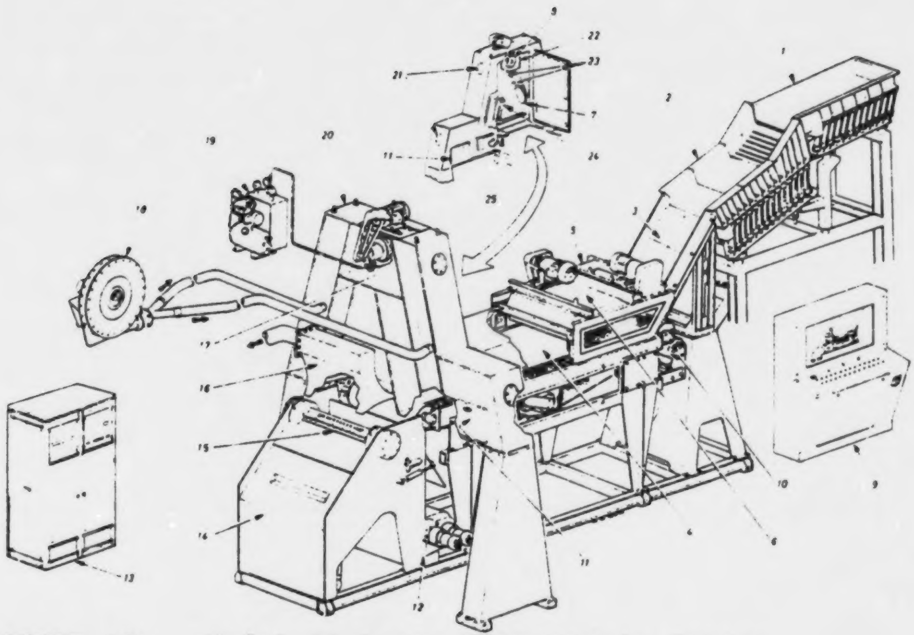


FIGURE 43: Model 16 Laser Photometric Sorter.

1 & 2 vibrating feeders. 3 slide plate. 4 main belt. 5 accelerating roller. 6 stabilising belt. 7 helium-neon laser. 8 rotating mirror drum. 9 control panel. 10 main belt brush. 11 Video rack. 12 black background assembly. 13 processor cabinet. 14 blast assembly. 15 blast manifold assembly. 16 extraction hood. 17 spindle assembly housing. 18 air curtain fan. 19 mirror air compressor and drier unit. 20 scanning cabinet. 21 cable entries. 22 perspex housing. 23 laser reference sights. 24 photomultiplier. 25 white standard.

size limit its capacity is 150 tonne/h. The MP 80 has few moving parts and as a result requires minimal maintenance. The feed material is vibrated from a surge hopper onto a 0.8 m wide belt, travelling at 2.1 m/s, to form a monolayer of spaced particles moving at the same velocity, (Figure 44). At the end of the belt the particles in free flight pass through a viewing zone which is brilliantly lit by quartz-iodine lamps. The lamps are housed in an overhead cabinet together with a solid state camera which is focused on the full width of the feed path. The sensing element of the camera consists of 1024 photosensors to give high resolution over the 800 mm width. As the mineral particle moves across the viewing area it is scanned one thousand times a second and a complete picture of the particle is built up and stored. This information is modified by an electronic filter and passed to the microprocessor for the decision-making process which is based on the spectral picture of the particle surface. From this a decision is made on whether to eject the particle. The microprocessor also evaluates the size of the particle and its position in the 800 mm wide stream. When the particle has to be ejected the microprocessor will: 1. select the correct ejector valve to coincide with the particle's position in the flight path, 2. control the time delay necessary for the particle to reach the ejection zone, 3. compute the duration of air jet so that it is appropriate for the particle size. One or more valves may be selected depending on particle size. The ejectors are high-speed, pilot operated spool valves, each capable of delivering 50 blasts of compressed air a second. The size and number of ejectors may be chosen although 16 or 32 are standard. The air required is 550 to 825 kPa (80 to 120 lb/in^2) depending on the feed to be sorted. When activated the air blast deflects the particle from its free fall path into a separate chute. Since the air blasts are directed downwards they tend to remove dust from the viewing zone and allow the optics to function at maximum efficiency.

The Oxylore process is an induced fluorescent sorting technique for the preconcentration of limestone, coal and oil shale.[219] The process utilizes selective adsorption of fluorescent coating to label either useful or refuse particles which can then be detected under ultraviolet light. The schematic diagram of the process is shown in Figure 45. The ore is first washed to remove fines and then sprayed with a water insoluble dispersion containing the

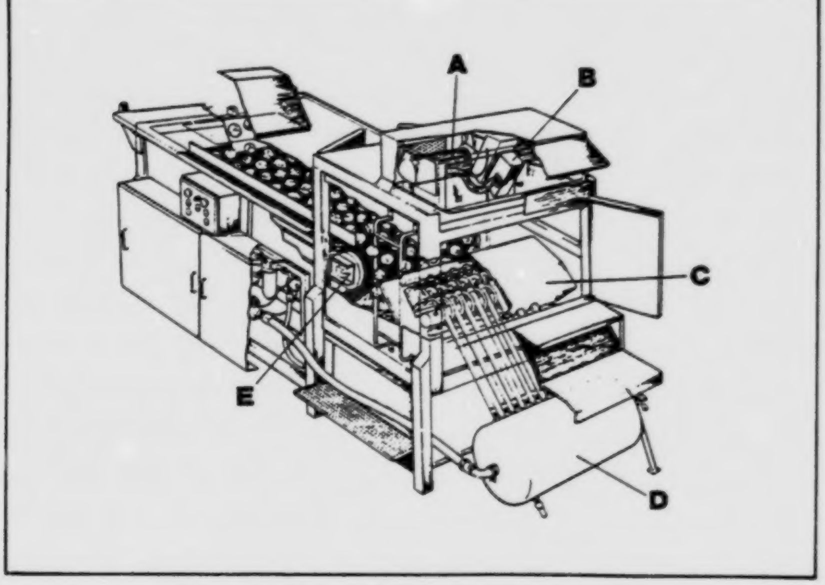


FIGURE 44: Line diagram of the Sortex MP80 machine. A. Quartz halogen lamps. B. Solid state camera.

fluorescent dye. The reagent adsorbs selectively onto particles and the excess is removed by wash water sprays. In the sorter, the feed is illuminated with ultraviolet light and during free fall, photosensitive cells detect fluorescing particles and activate water jets to deflect fluorescing particles from their trajectories. Photodetectors and water jets are located at 12.7 mm intervals. A larger particle will activate more than one jet and for a longer period to compensate for its greater mass. The feed size treated is within the 20 mm - 50 mm size range.

A unit of 118 t/h capacity has been built for pilot scale tests. When coal was sorted 58% ash reduction and 49% pyritic sulphur rejection were obtained at 81.1% clean coal yield, equivalent to 91.6% heating value recovery. In these tests the coal was labelled and thus became the ejected fraction. This sorting method may achieve good separation as long as the fluorescent dye is adsorbed selectively.

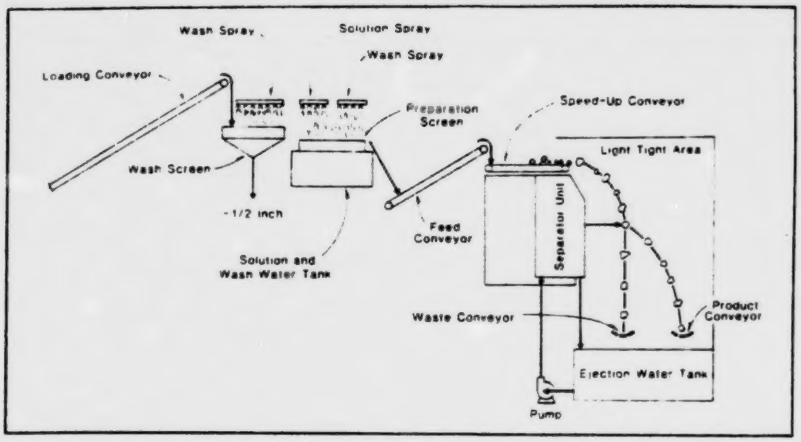

Wash Spray
Solution Spray
Wash Spray
Loading Conveyor
Preparation Screen
Speed-Up Conveyor
Light Tight Area
Wash Screen
-1/2 inch
Feed Conveyor
Separator Unit
Solution and Wash Water Tank
Waste Conveyor
Product Conveyor
Ejection Water Tank
Pump

FIGURE 45: Schematic diagram for induced fluorescence sorting.

For naturally radioactive minerals such as radium the gamma-ray emission of the particles may be detected and used in sorting. The RTZ Ore Sorters Mark VI A machine and Model 17 work on these principles.[220,221] When the uranium content of the ore is low, special techniques have to be used in which the scintillation count for each particle is accumulated and stored as the particle passes over several scintillation counters. General Mining Union Corporation, in cooperation with Gunson's Sortex, developed the RM161 machine operating on this principle.[222,223]

Neutron absorption separation has been used in the U.S.S.R. for the sorting of boron minerals.[224] In a neutron adsorption sorter the ore is delivered by a conveyor belt in between a slow neutron source and a scintillation neutron detector. The neutron flux attenuation by the ore particles is detected and used as the means of sorting. It is most applicable to the 25 mm to 150 mm size range. Boron minerals are easy to sort by neutron absorption since the neutron capture cross section of the boron atom is very large compared with those common associated elements and thus the neutron absorption is almost proportional to the boron content of particles. Photoneutron separation is recommended for the sorting of beryllium ores. The principle is that when a beryllium isotope in the mineral is exposed to a certain energy gamma radiation a photoneutron is released which may be detected by scintillation or a gas counter.

Outokumpu Oy has developed a new radiometric sorter which is available in modular form. The sorter is primarily intended for use as a preconcentrator and thus it is known as PRECON. Preconcentration tends to increase subsequent concentration capacity and to reduce reagent and energy consumptions. It may also save considerable transport costs when the ore has to be moved long distances to the concentrator. Each module contains two separation lines of 7 to 40 t/h capacity depending on the feed size. During separation the mineral particle, in free trajectory, is irradiated by two pairs of radioactive sources on either side of the detector which measures the scattered gamma radiation emitted from the particle. The signal is compared with a preset value and, as necessary, the particle is deflected from its normal trajectory by a blast of compressed air. Each detector has its own microprocessor which evaluates the grade of particle and controls the operation of the air jets. Ten to fifteen particles per second may be analysed in each line. Ore lumps from 35 mm to 150 mm with a size ratio of not greater than 2, may be separated on the machine. The metals assayed include Cr, Fe, Ni, Cu, Zn, Mo, Sn and Pb. The ore may be sorted into either two or three products. Power requirement of the PRECON is 6 kW per line. The size of the module is 8.7 m long by 2.5 m wide by 5.2 m high and it weighs 10 t.

A sorter which measures the electrical conductivity of particles and accepts or rejects accordingly was developed by Gunson's Sortex (Mineral and Automation) Limited.[225] The machine consists of a revolving disc, 1.5 m diameter, onto which the ore lumps are delivered by a vibratory feeder. The lumps are guided into a spaced single file at the periphery of the table. The conductivity of individual lumps is measured between two electrodes, the table and a contact brush. The average value of conductance for a predetermined length is computed. Comparison of the computed value with a preset value determines whether or not the particle should be ejected. Immediately after this inspection the lump interrupts a beam of light. If the lump has been sensed for ejection, the interruption of the light beam activates a high speed air valve, which blasts a jet of compressed air when the particle passes in front of the ejector nozzle. Pilot plant tests on 50 mm to 100 mm iron ore indicated a feed rate of about 25 tonnes/h. The unit occupies 4 m^2 of floor space and needs 2.5 m headroom. Electrical power requirements are about 2 kW and air consumption 7 m^3/min at a pressure of 550 kPa (80 lb/in^2).

The U.S. Bureau of Mines had developed sorters for use on native copper ores of Upper Michigan.[226,227] These sorters work on the metal detection principle. An excitation coil fed from an oscillator at a frequency of 125 kHz creates the radio frequency field below the transport belt. A pick-up coil above the belt is positioned with its axis at an angle of 90^o to the axis of the excitation coil. A conductive particle distorts the field and gives a signal through the pick-up coil. This measuring system is subjective to the size of conductive particles, as a coarser particle of the same conductivity gives a greater output.

A newly developed ore sorting system which is expected to expand the scope of sorting techniques to a wide range of previously unsortable minerals is offered by RTZ Ore Sorters.[228] The new machine, known as the Model 19 Conductivity/Magnetic Response sorter, remotely measures the electrical conductivity and magnetic characteristics of individual particles as they pass over the sorter. The net detectors employed are claimed to respond to even the slightest variation of electrical or magnetic properties, making the Model 19 sorter suitable for use on a wide variety of ores, including sulphides, oxides and native metals. The sorter is available in four different configurations to suit feed sizes from 25 mm to 150 mm, with capacities up to 120 t/h at the upper size limit.

Sorting is likely to have a role in the beneficiation of coal but optical sorting of coal from shale does not normally give adequate selectivity. Induced fluorescent sorting relies on the surface properties of minerals and, in certain cases, may not be selective enough. X-ray absorption should be more selective for ash rejection. Since the volumetric pyrite content of raw coal is relatively low, the X-ray absorption method is not useful for pyrite detection, but a combined system, using X-ray absorption for shale and back scatter for pyrite, may supply a detection method with good sensitivity. In such a combined system particles might be rejected at certain levels of ash or sulphur contents or according to a certain relationship of the two. If a sorter could be designed on this principle using a scanning detection system it would have high capacity and would not require spray washing of coal feed.

2.10 Flotation

Flotation uses the surface properties of mineral particles to effect separation. The natural surface properties of minerals are usually enhanced by absorbed reagents which make particles to be recovered in the froth product hydrophobic and the others hydrophilic. The hydrophobic particles adhere to air bubbles and are carried in the froth while other minerals remain in the pulp. Flotation is a very versatile process which is able to selectively separate a large number of minerals. However, flotation cannot treat coarse sizes. The upper size limit for industrial flotation of coal is around 1 mm.

There are several important differences between coal and mineral flotation. In mineral flotation, the entire feed is often finely ground and separated by flotation. Even when a coarser product is made by another separation method, a substantial fraction of the feed is still treated by flotation. In coal preparation, froth flotation is only used for the naturally broken fines fraction, whose mass and importance is negligible in comparison with the coarse product. Froth flotation was introduced into coal preparation partly to overcome fines disposal problems (blackwater) by recovering coal fines. The importance of flotation is growing as the fines content of run-of-mine coal is increasing. Another main difference is that minerals are much more constant in chemical composition than coal which is an extremely heterogenous material.[229] Coal is present in the form of macerals. Thus particles in a coal flotation circuit may occur as free macerals, composites of different macerals, composites of macerals and minerals, very fine free minerals which come from clay and coarser particles of shale. Apart from this, individual macerals do not have constant chemical compositions.

Thus, while mineral flotation involves the flotation of a fairly well defined phase, coal flotation is the separation of a number of macerals with varying flotation characteristics. Vitrain is more readily floatable than durain.[230] Low volatile coals are easier to float than most high volatile coals. Lignite is the least floatable form and anthracite is more difficult to float than high volatile bituminous coal. Coal flotation is adequately described by various authors.[229-232]

The froth process is the most important of the methods based on surface characteristics. The possible alternative separation methods based on surface activity are shown by Fuerstenau in Figure 46. Oil extraction involves the collection of hydrophobic particles into the oil phase. Since oil-solid intermolecular forces are much greater than air-solid intermolecular forces it is easier to collect fine particles at an oil-water interface than at an air-water interface. When the particles are extracted into oil droplets which are floated by air bubbles the method is called oil flotation or emulsion flotation. Spherical agglomeration is the collection of fine particles from an aqueous suspension into spherical aggregates within an oil phase. The agglomerates formed are simply screened out. Heterocoagulation occurs when particles of different minerals adhere to each other. Heterocoagulation is utilised in carrier flotation or ultraflotation. Here fine contaminants adhere to carrier particles added to the suspension. The carrier particles are floated taking the contaminants with them. Carrier flotation has been used industrially for the purification of kaolin. In agglomerate flotation agglomerates are formed and then floated with air bubbles. The rate of flotation of agglomerates is usually much higher than that of single particles. When flocs are formed selectively from fine suspended particles with polymer flocculants and these flocs are then floated, the process is referred to as flotoflocculation. In selective flocculation the selectively formed flocs are separated from the dispersed phase by sedimentation methods. Selective flocculation is an industrially established process for the concentration of non-magnetic taconites.

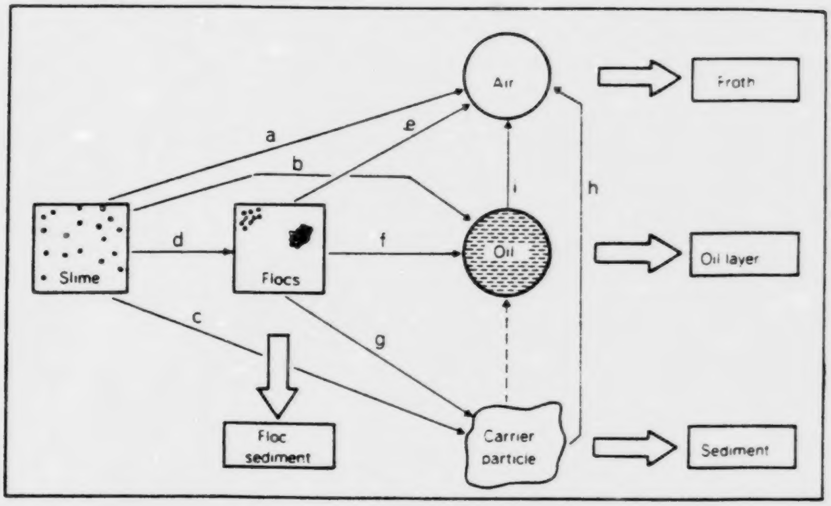


FIGURE 46: Schematic representation of separation methods based on surface activity. Froth flotation, a; oil extraction and/or spherical agglomeration, b, (d-f); Heterocoagulation, c; selective flocculation, d; Aggregate flotation, d-e; Ultraflotation, c-h, (c-j), (c-j-i); oil flotation, b-i, (d-f-i).

An enormous amount of research has been carried out on mineral flotation. Much of this has no relevence to coal flotation, but there is still a considerable amount which has direct significance. A selection of the appropriate areas will be summarised here, together with some related work on coal.

2.10.1 Flotation Reagents

Sutherland and Wark distinguished four functional classes of flotation reagents: collectors (or promoters), frothers, activators and depressants.[234] Activators and depressants are also referred to as modifiers. The function of a collector is to adsorb onto the mineral to be floated and make its surface hydrophobic so that it will adhere to air bubbles. Collectors must act selectively. Frothers increase froth stability by adsorbing at the air-water interface. They permit the removal of mineralized froth by extending froth life. An activator selectively adsorbs onto mineral surfaces and facilitates the adsorption of the collector on a certain mineral by itself first adsorbing on the mineral. Collectors and frothers have been used in coal flotation but activator and depressant usage is more recent and is usually restricted to sulphur reduction.

Many coals are easily floated with frother only, due to their inherent hydrophobicity. For lower rank and oxidized coals the addition of hydrocarbon oil is often necessary. Efficient conditioning of coal slurry with a sufficient amount of the proper type of collector is important for high clean coal recovery. The best collector-frother combination for the actual raw coal should be carefully evaluated. As some case studies show, considerable savings in reagent cost and increased recovery may result from the correct choice of reagents.[235]

The basic function of a frother in flotation is to provide the proper type and quantity of froth. Froth is stabilised by positive adsorption of frother molecules at the air/water interface. The increased concentration of frother molecules present in the interface reduces surface tension to slightly below that of pure water. Any disturbance of the froth which will cause stretching of the froth filament will reduce the concentration and simultaneously increase surface tension. The increased surface tension will cause the froth filament to contract, so increasing its stability. This is a dynamic process and therefore the time of diffusion of frother molecules has to be of the order of the time required for local film flow. Low molecular weight frothers diffuse too rapidly to the surface to allow time for film flow. On the other hand high molecular weight frothers may not produce sufficient surface tension change. Three phase froth with inter-filament mineral load tends to increase froth stability.

But frothers do more than just increase froth stability.[236] [237] The second important, but less well known, frother function is regulating the rate of flotation, which occurs through interactions with the collector both in the liquid phase and on the mineral surface. The flotation rate controlling function of a frother is explained by the adsorption of the froth-collector complex molecules onto the mineral surface where they serve as nucleation sites for bubble formation. These micro-bubbles formed by gas precipitation or cavitation facilitate the adhesion of bubbles large enough to float particles and thus control the rate of flotation. Indeed, numerous experimental observations support the interaction of frother with the collector. Operators know that the flotation rate can be varied more easily by increasing frother dosage or by changing frother type than through alterations to the collector. It has been demonstrated experimentally that flotation is possible when a non-surface active frother is used. The presence of an inactive frother improves mineral recoveries compared with use of collector alone, although often no increase in contact angle is apparent. Other workers found that froth volumes produced with no mineral present were small with only the addition of collector and greater volumes were produced with frother. When both collector and frother were present considerably greater froth volumes occurred than would be expected from the sum of the individual froth volumes, suggesting molecular association in the liquid phase. Since solids were not present the excess froth could not be a result of stabilisation due to mineralised froth. It is recognized that the highest mineral recovery occurs at certain concentrations of the collector and frother. Any variation either way from these dosages reduces recovery. Furthermore, frother concentration appears to be more critical than collector concentration. This phenomenon is acknowledged by some plant metallurgists who try to maximise recovery by the careful choice of collector and frother additions. The optimum condition is also beneficial since here the recovery is not sensitive to small changes of frother and collector concentrations, as optimum conditions occur on the top of the recovery-frother concentration-collector concentration surface. The frother-collector interaction may be explained by the diffusion of collector molecules through the adsorbed liquid film into the mineral particle. This film is stabilised by a network of frother molecules in the vicinity of the mineral surface. It is thought probable that the collector and frother molecules are attracted to each other by van der Waals forces.

2.10.2 Factors affecting flotation

2.10.2.1 Particle Size

It has long been known that intermediate size particles have the highest flotation rates and that coarse particles and fines are recovered at lower rates. The low flotation rate of small particles may be explained by their low mass and insufficient kinetic energy to cause particle-bubble attachment and by the predominance of electrical repulsive forces. Coarse particles have lower rates of flotation as they require greater forces to be carried into the froth, for which ideal conditions are needed. Here, we have considered particle recovery by ideal flotation. Trahar distinguishes two main mechanisms of recovery: by flotation and by entrainment.[238]

In recovery by flotation the mineral particles are carried into the froth by attachment to air bubbles and thus this process is usually selective. In recovery by entrainment both floatable and non-floatable mineral particles, particularly fines, are carried over in the froth lamellae to the froth product. This process is not selective. In the fine and intermediate regions the rate of recovery by flotation increases sharply with particle size as the flotation rate is proportional to particle diameter raised to a power between 1.5 and 2. For recovery by entrainment the rate of flotation increases towards the finer sizes. So for the fine particle size region flotation recovery is almost entirely by entrainment, which is a nonselective process. Trahar showed conclusively that the amount of fines floated by entrainment was directly proportional to the water recovered in the froth product.

Hemmings developed a conductivity probe for the measurement of liquid film thickness. In this probe alternating current is passed through two platinum electrodes and the resistances of pulp and liquid film are measured. From the ratio of resistances and from the length of electrode the thickness of the froth lamella may be calculated. The froth lamella thickness was measured in copper-lead-zinc and tin concentrators and in a coal preparation plant. In the concentrators, thicknesses ranged between 50 μm and 250 μm while in the coal preparation plant they varied between thicknesses of 280 μm and 830 μm.

These values emphasised the importance of carry-over of entrained fines by the froth lamella liquid when treating fines, as the carry-over drastically limits the grades of concentrates.

An alternative view of the flotation behaviour of ultrafine particles is given by Hemmings of MD Research Co., Australia.[240] The ineffectiveness of the flotation process for the beneficiation of finer than 10 μm particles has previously been attributed to: 1. Low probabilities of collision and adhesion of ultrafines to air bubbles, lowering the rate of flotation. 2. Surface chemical differences as a function of particle size, resulting in greater similarity between the surface properties of ultrafine values and gangue, thereby reducing the selectivity of the process. 3. Mechanical entrainment of gangue in the froth, reducing concentrate grade. According to Hemmings the first two assumptions have not been substantiated in industrial flotation. He gives an alternative explanation for ultrafine particle behaviour in flotation based wholly on froth related phenomena, supported by experimental evidence. His conclusions are derived from calculations relating to froth lamellae and batch flotation tests on a scheelite ore from King Island, Tasmania. He explains fine particle behaviour in terms of liquid-lammella thickness and froth carrying-power. When the mass of solids entrained in the froth lamella is compared to the mass of particles floated on the two surfaces of the lamella, it becomes clear, that for fine particles the entrained mineral mass is large and the floated mass is relatively small. The consequence is low concentrate grade because too much of the suspension is carried over non-selectively in froth lamallae. Furthermore, as the same froth area can carry a much greater mass of coarse particles than fine, the rate of flotation is lower for fine particles. In fact, the carrying power of the froth determines the rate of flotation. Hemmings gives the following operating requirements for the effective beneficiation of ultrafine particles: 1. Mineral particles must be hydrophobic and gangue minerals must be hydrophilic. 2. The zeta potential of minerals must be moderate to high and of like sign, otherwide heterocoagulation will occur and selectivity will suffer. 3. Surfaces of froth lamellae should bear the maximum density of particles while the film thickness is maintained as low as possible in order to increase flotation rate and reduce gangue entrainment.

The poor flotation response of many coals has often been attributed to rank, degree of oxidation, petrographic composition or slime coating. Undoubtedly these factors are important in coal flotation but current work shows that the size distribution of the feed may be of overriding importance. Losses of relatively coarse, plus 250 μm coal which is normally considered easily floatable can be very substantial.[241,242] Two stage collector addition is proposed to improve results substantially and refloating the first stage flotation tailing should further improve results. Using the same coal, single stage flotation produced a 59.6% yield at 11.6% ash content, two stage collector addition improved results to 79.6% yield at 11.0% ash while reflotation gave 81.0% yield at 9.9% ash content.

When the particle size is reduced the specific surface area of the particle becomes large and the particle mass diminishes. The surface and electrochemical properties of fine particles also tend to be different from the properties of coarse particles of the same mineral. The consequences of the high surface energy, high specific surface and small mass of fine particles in flotation are given schematically by Fuerstenau in Figure 47. Due to their small mass, fine particles tend to be carried into the froth, entrained in the liquid or mechanically entrapped amongst particles being floated. When such particles are gangue minerals they reduce the grade of the concentrate. The large specific surface area of fine particles accounts for their capacity for reagent adsorption. A significant proportion of the reagent is consumed by small particles and this may not leave sufficient reagent for the flotation of larger particles, thus reducing their recovery. Calculations indicate that most of the surface area of even a minus 13 mm crusher product is in the minus 2 μm fraction although this fraction accounts for only 0-13% of the total weight. Finer mineral particles have higher specific surface energies because of their increased number of edges, corners and crystallographic imperfections. The high specific surface energy of fine particles influences flotation in various ways. Increased dissolution from particle surfaces introduces undesirable impurities into the solution and affects collector-mineral interactions. Dissolved cations may cause unintentional activation of minerals which should remain in the pulp or may cause depression of minerals which are intended to float. Rapid oxidation of fine sulphide particles may prevent their successful flotation. The high specific surface energy of fine particles also increases their tendency to adsorb collectors non-specifically and thus causes flotation of undesirable minerals.

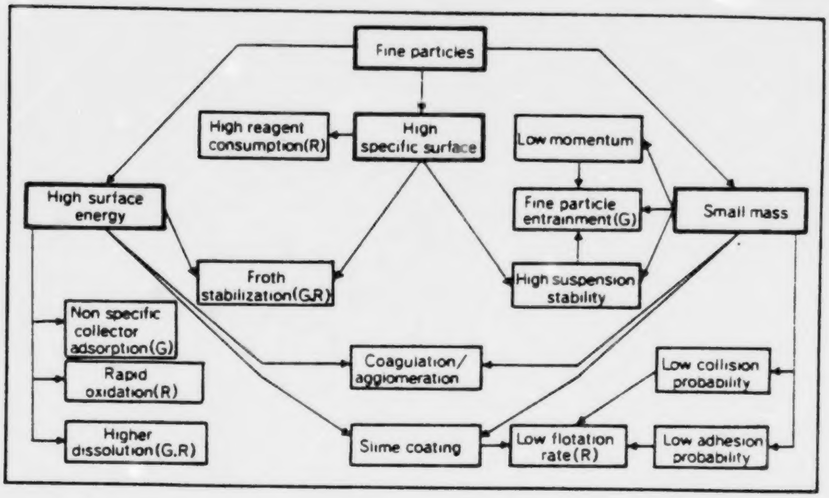

Fine particles
High reagent consumption(R)
High specific surface
Low momentum
High surface energy
Fine particle entrainment(G)
Small mass
Froth stabilization(G,R)
High suspension stability
Non specific collector adsorption(G)
Rapid oxidation(R)
Coagulation/ agglomeration
Low collision probability
Higher dissolution(G,R)
Slime coating
Low flotation rate(R)
Low adhesion probability

FIGURE 47: The relationship between properties of fine particles and their flotation behaviour. (G) and (R) show effect on grade and recovery.

2.10.2.2 Oxidation

When coal is exposed to air and moisture it will undergo a series of physical and chemical changes, known as oxidation. Vitrain is more readily oxidized than other macerals.[243] Oxidation reduces the water repellancy of coal and, consequently, its floatability. The negative value of zeta potential of coal increases with oxidation and the coal slurry pH decreases.[244] At low pH values, Fe, Ca, Mg and Al ions are released from the coal into the aqueous slurry. These ions are undesirable for flotation. When the coal becomes oxidized, microfractures occur on its surfaces, making particles weak and more friable. The surface area also increases and thus the coal requires more reagent for flotation. Therefore, whenever possible coal oxidation should be prevented. Some coals are mined in an oxidized condition and, to some extent, the oxidation can be remedied by appropriate admustment in reagent types and dosages.[245]

When raw coal fines are difficult to float froth conditioners may provide the answer to the problem. A study by members of the Dowell Division of Dow Chemical, U.S.A. suggests that the natural oils on coal surfaces are necessary for efficient flotation.[246,247] Where these oils are absent or have been lost, froth conditioning techniques can be applied to improve recoveries. Froth conditioners were developed to enhance the flotation of highly oxidized coal but have also been found useful for difficult-to-float freshly mined coal. Coal characteristics greatly influence flotation as coals with high oxygen to carbon ratios are usually difficult to float.

Froth conditioning also increases flotation rates and recoveries of coarse particles. In flotation tests on numerous varieties of coal, Dowell's M210 froth conditioner increased yields and frequently reduced the clean coal ash content. Typically, the M210 is added at 0.1 kg/t dosage as a reagent blend of 9.5 parts diesel oil, 4 parts frother and 1.5 parts M210. Addition of froth conditioner may also reduce the necessary diesel oil and frother dosages.

Prereduction of oxidized coal with SO_2 and subsequent flotation improved ash rejection without affecting clean coal yield.[248] The frother/collector (Montanol 361) requirements were higher when the SO_2 dosage was increased. At higher SO_2 dosages it was possible to reduce clean coal ash to 7% at yields between 85% - 90%. However, up to 2.5 kg/t frother/collector was required to maintain adequate froth stability.

2.10.2.3 Slime Coating

Slime coating occurs when fine particles adhere to larger particles due to long range intermolecular forces. Slime coating can be detrimental in several ways. When fine particles of the valuable mineral coat larger gangue particles, the gangue may float and reduce the concentrate grade. On the other hand, when fine gangue particles coat coarser particles of the valuable mineral the coating will prevent attachment of air bubbles and reduce the recovery significantly. Reduction in coal flotation yield combined with increasing clean coal ash content is often experienced when large amounts of clay are present in the flotation feed. The fine clay adheres to the coal particles forming "slime coating" and prevents the attachment of air bubbles to coal particles. According to Firth and Nicol, the presence of clay in the feed does not necessarily result in slime coating and adverse flotation conditions, as clay particles with clean surfaces do not adhere to coal particles.[249] However, when clay particles are contaminated with humic acid or oily substances, slime coating results. At high pH values, humic acid and oil collector droplets will not adsorb onto shale surfaces and therefore will not cause slime coating. At low pH values the converse is true. These findings have significant implications for coal flotation circuits operating with acidic water, produced, for example, by the oxidation of pyrite within the feed material. Thus pH control in such circuits may improve selectivity.

Clay minerals, such as illite, montmorillonite and kaolinite are common constituents of coal seams or adjacent strata. These minerals are objectionable in flotation as they cause slime coating, induce overfrothing, reduce flotation rates and increase clean coal ash content.[245] Since the slimes adsorbed on the coal surface may be dispersed by appropriate electrolytes, it is probable that their adsorption is due to electrostatic forces. The electrokinetic potentials of illite, montmorillonite and kaolinite are negative between pHs 2.5 - 11.0 and thus the clay particles should not absorb on the negatively charged coal. As clay platelets are negatively charged on their flat faces and positively charged on their edges, a negative overall electrokinetic potential results. The clay particles, therefore, can adsorb with their edges onto coal particles. The various ions present in the water also influence the slime coating.

2.10.2.4 Electrokinetic Potential

Electrokinetic measurements have been widely applied to minerals and detailed interpretations of the influence of electrokinetic potential on collector adsorption have been published.[250-256] Due to the heterogenous nature of coal, the interpretation of electrokinetic measurements is more difficult. The application of electrokinetic measurements to coal flotation is important as it helps to clarify the actions of electrical surface forces and aids the solution of flotation problems such as those occurring in collector adsorption, oxidized surfaces and slime coating.

The zeta potential variation, with pH, of coal from No. 26 colliery, Devco, N.S., has been studied.[257,258] The isoelectric point for this coal was pH 5.1, below which the coal had positive electrokinetic potentials. The isoelectric point of the bright component of the coal (vitrain and clarane) was pH 4.2, while that for the dull component (duraine) was pH 6.1. Hypothetical models for the mechanism of surface charging are given.

A study of electrokinetic behaviour of oxidized coal indicates that it is feasible to separate oxidized coal from shale and pyrite by amine flotation.[259] It was found that increasing oxidation increased the negative value of the electrokinetic potential and decreased the isoelectric point. Flotation recovery increased at lower pH values.

The maximum hydrophobiscity of the oxidized coal, with dodecylamine collector occurred near the isoelectric point of pH 6.5, at which pH value ferrous sulphate depressed pyrite effectivly. In another electrokinetic study on fine and oxidized Kentucky coal a cationic and an anionic collector were tested.[260] The catronic collector readily adsorbed on the negatively charged coal giving good recoveries, but the anionic collector did not adsorb on the coal and thus flotation was impossible.

2.10.2.5 Other factors

The beneficial effects of the inorganic salts on frothing have long been known byt they have not been exploited in flotation practice. Yoon has carried out coal flotation tests with the addition of 1% to 3% inorganic salts and reports that coal may be floated even without collector and frother.[261] In other tests, he generated a dispersion of microbubbles in water, which resembled milk, and which he used in flotation. The clean coal yields were high but ash rejection was poor because ash particles were mechanically entrained in the froth product. Two stages of flotation were needed to reduce the ash content. The generation of microbubbles requires high frother dosages. Perhaps a combination of both techniques, i.e. using small amounts of salts with micro-bubbles may reduce frother requirements. However, the salt chosen must not be harmful if any remains on the clean coal.

The significance of pulp potential was demonstrated in laboratory tests some time ago although its exact meaning when applied to mixed mineral suspensions is not well defined. Pulp potential is measured as the electrical potential occurring between a platinum electrode and a saturated calomel electrode, both immersed in the pulp, which is then converted to standard hydrogen electrode potential by deducting the potential difference between the saturated calomel electrode and the standard hydrogen electrode which is 245 mV. In a pulp containing various minerals the platinum electrode is expected to adopt a potential near that of the conducting solids components present in the pulp. Negative pulp potentials are associated with reducing conditions and positive potentials occur in oxidising conditions. At the lead-zinc concentrator of Mount Isa Mines Ltd., Johnson and co-workers carried out a detailed study of the effects of pulp potential on galena flotation.[262] Their study is the first full plant scale work to demonstrate the detrimental effects pulp potential may have on flotation. They found that the flotation rate of galena was low in the first few flotation cells for all sizes, but as gradually more oxidizing conditions occurred due to aeration, the pulp potential and the flotation rate increased. Thus it appears that pulp potential is an important variable whose effect on the flotation of various minerals should be studied. Pulp potential is not difficult to monitor in a flotation circuit and perhaps it could provide additional useful information for computer control of the mineral recovery process. For oxidized coal, reducing conditions associated with negative pulp potentials may give better conditions and increased flotation rates.

Intermittent aeration of flotation machines has been proved theoretically and experimentally superior to continuous aeration.[263] Intermittent aeration may consist of alternating on and off periods or of high and low aeration rate periods. This type of aeration not only reduces energy requirements but also increases the upper size limit of floatable particles. This patented method introduces another variable to the flotation process and should improve the design of large volume flotation machines. The results of tests carried out on fluorite, potash and quartz flotation appear to support the designers' claims.

2.10.3 Split Conditioning

As with most mineral separation methods, flotation also gives highest efficiency on relatively narrow feed size ranges. The conditions required to float various sizes are different. "Split conditioning" is the separation of a feed slurry into two size fractions followed by separate conditioning and flotation of the size fractions. The size separation is usually carried out by hydrocyclones at about 150 μm. Split conditioning gives superior performance for both size fractions. The method was advocated by Chang and Dasher in 1959 when they published a paper on split conditioning and fines flotation used at the Crucible Mine Pennsylvania.[264] The flow diagram of this plant is given in Figure 48. The process was granted a U.S. patent, No. 297989, which has now expired, but today the process may still be regarded as an advanced technique for fines treatment.

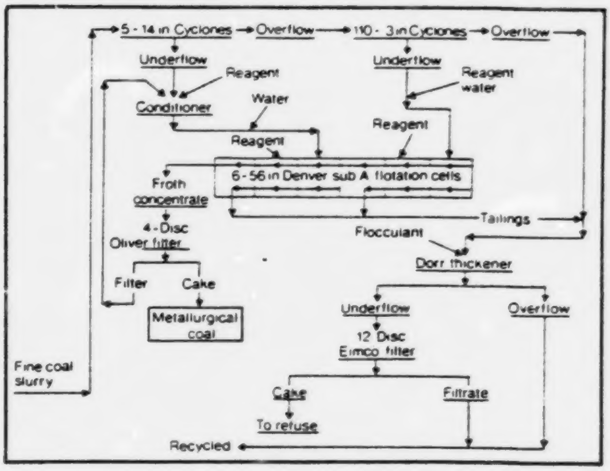

5 - 14 in Cyclones
Overflow
110 - 3 in Cyclones
Overflow
Underflow
Reagent
Underflow
Reagent
water
Conditioner
Water
Reagent
Reagent
6-56 in Denver sub A flotation cells
Froth
concentrate
Tailings
4-Disc
Oliver filter
Flocculant
Dorr thickener
Filter
Cake
Metallurgical
coal
Underflow
Overflow
12 Disc
Eimco filter
Fine coal
slurry
Cake
Filtrate
To refuse
Recycled

FIGURE 48: Flow diagram of the Crucible fine coal plant using split conditioning in 1959.

A flotation circuit which employs split conditioning in the cleaning of a highly pyritic coal is shown in Figure 49.[265] The solids concentration by weight of the coarser hydrocyclone underflow is adjusted to 15% before conditioning and that of the finer cyclone overflow to about 6%. Two conditioners are used here, in the first ferric chloride is added to depress pyritic sulphur and in the second conventional coal flotation reagents are added. Both flotation circuits have a rougher bank and a cleaner bank of flotation cells. The flotation tailings are flocculated, thickened and filtered and the water is recirculated to the plant.

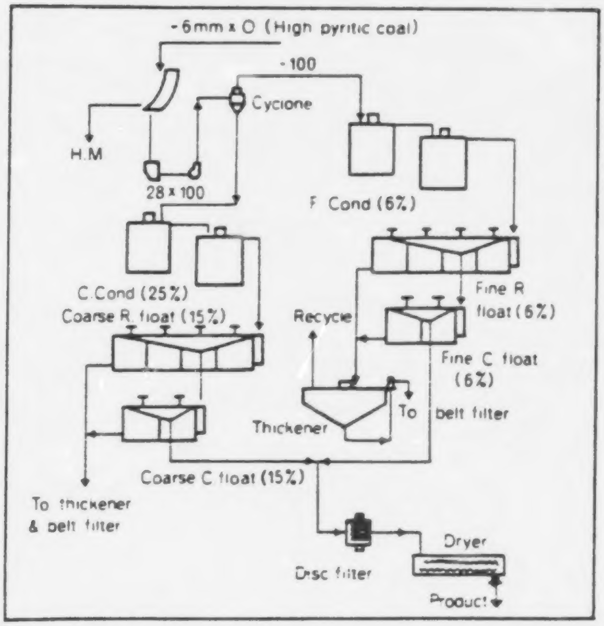

-6mm x O (High pyritic coal)
-100
Cyclone
H.M
28 x 100
F Cond (6%)
C.Cond (25%)
Coarse R. float (15%)
Recycle
Fine R float (6%)
Fine C float (6%)
Thickener
To belt filter
Coarse C float (15%)
To thickener & belt filter
Dryer
Disc filter
Product

FIGURE 49: Flotation circuit for treating highly pyritic coal.

2.10.4 Flotation Machines

An extensive and critical review of the operating principles and characteristics of flotation machines from various manufacturers available throughout the world is given by Young,[266] (summarised in Mineral Processing, Mining Annual Review, 1982, pp. 259-263). A broad survey of flotation reagents, machines and processes based on U.S. patent literature is given by Ranney.[267] Detailed discussions of design principles are given by Harris[268] and Barbery.[269]

Flotation machines in current industrial use may be divided into four categories:[266]

1. Mechanical machines. These are characterised by a mechanically driven impeller, which agitates the pulp and disperses air bubbles into it. When aeration is achieved purely by the impeller, the machines are called "self-aerating" and when the machines receive air from a blower they are referred to as "supercharged". According to pulp flow, machines incorporating weirs between cells are known as "cell to cell", otherwise as "open flow".

2. Pneumatic machines. They have no impeller but rely on compressed air to aerate the pulp.

3. Froth separators. In these a froth bed is utilised for the separation of the preconditioned feed pulp which is fed onto the top of the froth bed. Hydrophilic particles pass through the froth bed while hydrophobic ones are retained and removed in the froth product.

4. Flotation columns. These make use of the countercurrent flow of bubbles and slurry. Gangue slimes are often removed from the froth by wash water.

Mechanical and pneumatic machines were the first types to be used, whereas froth separators and columns have only been known for the last 20 years. Mechanical machines have dominated the industry, particularly in the West. This dominance is not necessarily due to the design excellence of the mechanical machines but rather to their availability and to customer resistance to change. Pneumatic machines are rarely seen today apart from the Davcra cell which is a special type. Columns and froth separators have yet to gain wide acceptance; they are used far more in the U.S.S.R. than elsewhere.

The different types of flotation machines all have their own individual characteristics. Froth separators are ideal for coarser separation but are not very selective on fines. Flotation columns with froth washing may greatly improve the rejection of gangue fines. The additional mechanical barriers of the Leeds Column further enhance this rejection. Perhaps in the near future, mechanical flotation machines will cease to be the only type considered for installation and more use will be made of columns and froth separators. The flotation plant of the future may even be designed to utilise the inherent advantages of various types of machines for different duties leading to a decline in the dominance of mechanical flotation machines.

2.10.4.1 Mechanical Machines

Mechanical machines are the most commonly used type and they are made in a wide range of sizes. The development of larger sized flotation machines which started in the 1950s is still progressing. The motivation for the design of larger and more efficient flotation machines came from increased capital, operating and energy costs and from the steadily decreasing ore grades. More recently, the low metal prices necessitated the lowering of production costs to allow companies to remain economically viable.[270] Today, most flotation machine manufacturers offer large capacity flotation cells which promise savings in both capital and operating costs. The largest cell Outokumpu Oy made in 1973 was of 16 m^3 while by 1979 their largest cell was 38 m^3 capacity.[271] They have developed a new rotor which has separate slots for air and slurry, acting like a slurry pump and giving very effective mixing. The U-shaped tank used minimises the settling out of solids in the machine. In the 1960s Wemco made flotation machines up to 8.5 m^3, in 1978 they introduced 28 m^3 cells and in 1982 they developed a 42.5 m^3 machine.[272,273] Large Dorr-Oliver cells now offered in sizes up to 71 m^3 also have U-shaped tanks.[274,275] The Dorr-Oliver impeller mechanism runs at a relatively low peripheral speed and reportedly has very high power efficiency. Galigher manufactures AGITAIR flotation machines up to 42.5 m^3 capacities.[276] Properly designed large flotation machines can have a significant impact in reducing power costs. They occupy less floor area and building volume, have fewer parts, require less supervision, cost less per unit volume and consume less power/t of feed treated than smaller flotation machines. The decreases in floor area, first cost, power cost and number of agitator mechanisms as compared with a 2.8 m^3 cell is shown in Figure 50. From the graphs it is apparent that great reductions of these cost related factors may be achieved by selecting larger cells. However, the number of cells in a bank should not be reduced below a certain minimum to prevent short circuiting of the pulp which reduces recovery.

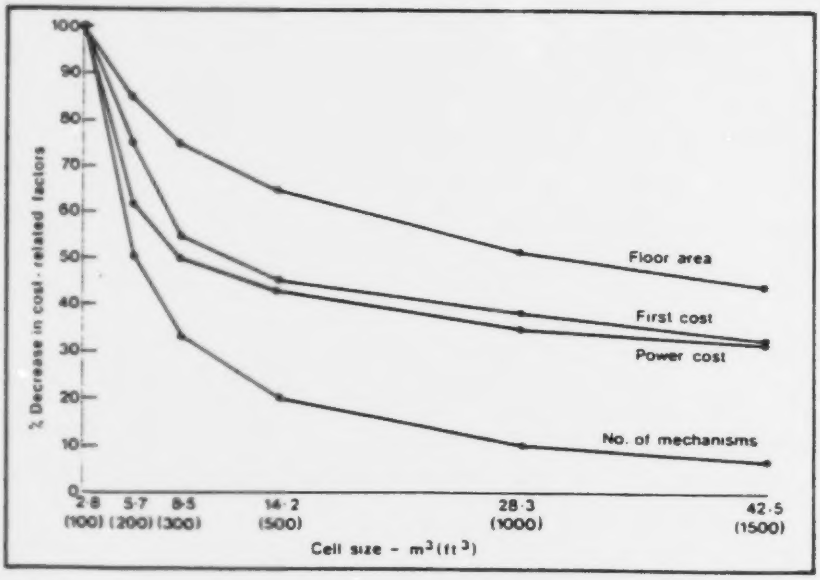


FIGURE 50: Economic advantages of large flotation cells.

The optimisation of the hydrodynamics of the flotation process, especially for large volume machines, has received considerable attention recently.[277,278,279] In scaling up flotation equipment the effects of many variables have to be optimised. There are four distinct hydrodynamic regions in these machines: mixing, transport, separation and froth removal regions. These regions differ significantly in the amount of turbulence present, ranging from highly turbulent flow conditions in the mixing region to smooth laminar flow in the froth removal region. According to Plouf of the Joy Manufacturing Co., a flotation machine that is designed for optimum capacity, high mineral grade and recovery and low power consumption should contain the following: 1. A controlled airflow. 2. A means of converting the supplied airflow into fine bubbles. 3. A mixing region which maintains suspension of the particles in the liquid. 4. Sufficient depth and volume of the mixing and transport regions to allow for high probability of mineral particle/bubble contact. 5. A quiescent separation region to facilitate separation of mineral laden froth from gangue particles. 6. A froth removal region permitting drainage and removal of froth.

The double or multi-wobble impellers are said to be a considerable improvement in impeller design.[280] They greatly enhance the performance of cells over 8.5 m^3 capacity. In coal flotation tests, throughput capacities doubled and clean coal yields increased by 10%.

Maxwell cells are not true mechanical cells as essentially they are cylindrical conditioners adapted for flotation. The mechanism is an unbaffled radial, back-swept, flat bladed turbine. Air is blown into the cell through a vertical pipe beneath the impeller. The tank contains one or two internal froth launders which connect to the froth outlets by radial launders. Maxwell cells offer outstandingly low installation, power and maintenance costs. They use very much less air than other designs. Maxwell cells are available with capacities up to 57 m^3.

In coal flotation the use of mechanical flotation machines predominates, although pneumatic machines are also used.[265] Two major tank designs are employed: 1. The flow-through type, in which the pulp flows through the bank of cells freely. 2. The cell-to-cell type, in which the pulp has to pass over an adjustable weir between cells. In Europe the cell-to-cell design is preferred for its higher selectivity, while in North America the flow-through type is favoured for its higher throughput with little sacrifice of recovery.

2.10.4.2 Pneumatic Machines

There are few types of pneumatic cells in current use. The Callow and MacIntosh types are hardly used today. The Forrester matless-airlift machine is still used in a few older mineral processing plants. The Heyl and Patterson Cyclo-Cells achieve aeration by submerged vortex chambers which discharge a high velocity jet of pulp and air in the form of a hollow cone.[281] Air is supplied at a pressure 150 kPa (2 psi). Cyclo-Cells are available from 3 m - 10 m long and 1.5 m - 2.4 m wide, treating up to 15t/h feed solids per cell.

The Davcra cell was developed in 1970 by Zinc Corporation in Australia. The cell uses an injection nozzle to mix pulp and air. The nozzle directs swirling concentric jets of pressurized pulp and compressed air into the separator tank, (Figure 51). As a result fine air bubbles are generated.

The cell uses a deep froth layer with a direct froth overflow. A 360 t/h Davcra cell was installed at Palabora Mining Co., South Africa, as a copper rougher. The cell performed well but some difficulties were experienced with nozzle wear.

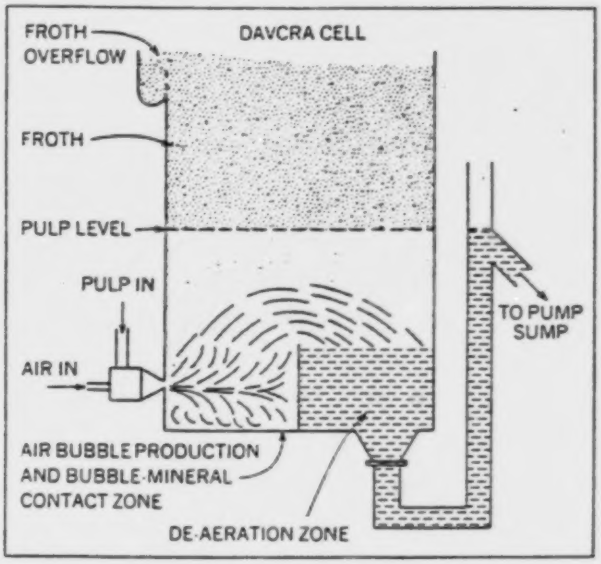

FROTH OVERFLOW
DAVCRA CELL
FROTH
PULP LEVEL
PULP IN
AIR IN
TO PUMP SUMP
AIR BUBBLE PRODUCTION AND BUBBLE-MINERAL CONTACT ZONE
DE-AERATION ZONE

FIGURE 51: Schematic diagram of the Davcra cell.

At the Technical University of Clausthal, Essen, F.R.G., a new type of flotation cell was developed in which aeration and particle-bubble attachment take place in one vessel and the separation of froth from slurry in another.[282] The separation of the two functions is advantageous because they require different hydrodynamic conditions. The slurry is aerated with fine bubbles in a specially designed unit in which particle/bubble attachment occurs within a few milliseconds after bubble formation. Freshly formed air bubbles are known to have greater probability of attachment. From the aeration unit the slurry passes to a 28 m^3 cylindroconical tank for the separation of froth from slurry. Here, the froth overflows into an annualar launder, and the tailings slurry is removed through the apex of the cone. Feed rates between 500 m^3 and 900 m^3/h have been achieved with slurries containing minus 100 μm coal, conditioned with a collector. The clean coal had 7% to 8% ash content and the refuse contained 68% to 78% ash.

2.10.4.3 Flotation Columns

The flotation column was invented in Canada in 1961 by Boutin and Tremblay. Conditioned ore is introduced into a tall column about a third of the way down, air is blown in through a diffuser at the base and wash water is fed in at the top of the column. Two counter current flows occur. In the lower zone, which is the recovery section, the descending ore particles encounter rising bubbles. In the upper section, the rising froth meets the down flowing wash water which washes down the fine gangue. Column Flotation Co. of Canada Ltd., has recently started to supply units for commercial use. The first plant installation was at Mines Gaspé Division of Noranda Mines, Quebec, in June 1980, for the flotation of molybdenite from the copper-molybdenum bulk concentrate.[283,284] Before the installation of flotation columns the circuit consisted of six flotation banks with 14 stages. Long retention times and large circulating loads between banks were characteristic of the operation. Eight cleaning stages were replaced by two columns, one of 950 mm diameter and the other of 460 mm. The introduction of these columns resulted in marked improvement of molybdenum recovery. In the same plant an 1,830 mm diameter unit has now replaced the mechanical roughers.[266] The same sized column is used at Brunswick Mining and Smelting as a zinc cleaner. An 1,830 mm diameter column is capable of processing 2,000 t/d of a typical sulphide ore. The Canadian column uses reduced amounts of frother to control bubble size and to reduce the depth of froth bed. In ideal operating conditions minimal froth bed is desirable as it mechanically entrains fine gangue. Without froth the fine tailing material is washed back from the wake of air bubbles resulting in a high grade concentrate product.

Tests were carried out on a high sulphur coal from Devco, N.S., using a 50 mm pilot plant flotation column.[285] This column was able to operate at unusually high solids concentrations of up to 32.5% by weight. The reverse flotation of pyrite from a coal concentrate of finer than 0.3 mm was quite successful, rejecting 80% of the pyritic sulphur.

The Flotaire flotation column was designed by Hollingsworth and Sapp for the treatment of Florida phosphates and is manufactured by Deister Concentrator Co. Inc., (Figure 52). It is able to float phosphate particles up to 1.2 mm. This column does not use porous air diffusers, but high pressure water (about 280 kPa) containing frother induces air through aspirators. The air-water mixture is fed into the base of the column below punched plates and flows upwards as a uniform stream across the total tank area. Flotaire columns are made from 0.13 m^3 to 19 m^3 capacities with heights of 4 m to 4.5 m. They do not use wash water but rely on froth bed processes for the rejection of entrained gangue particles.

Polish and Russian flotation columns are described in a book by Tyurnikova and Naumov.[286] Essentially these designs employ deep froth beds to give clean concentrates. Where wash water is used it always appears to be in conjunction with a froth bed.

The Leeds Column was invented by Dell at Leeds University and is now made under licence at Norton-Harty Colliery Engineering Ltd. for application in the coal industry.[287,288] The Leeds Column is essentially a number of froth cleaning barriers which perform a multi-stage cleaning action, (Figure 53). The identical barriers are vertically spaced at about 230 mm intervals. Each barrier consists of two sets

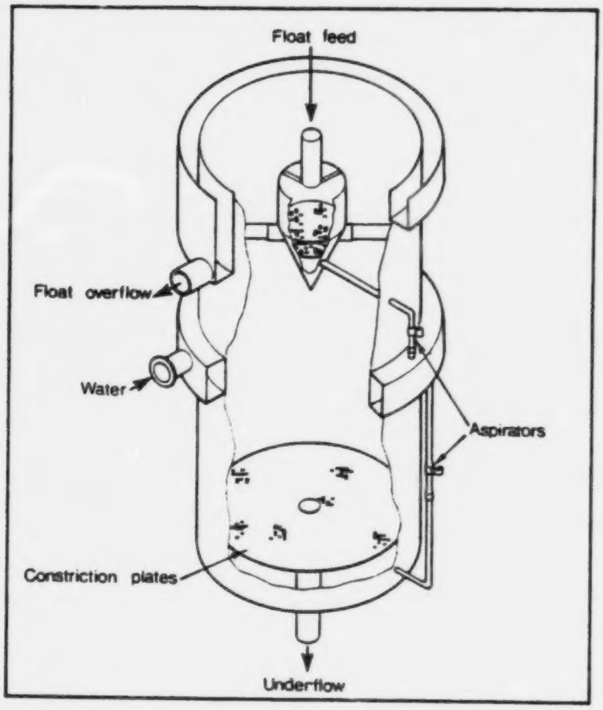


FIGURE 52: General arrangement of the Flotaire flotation column.

of parallel round bars arranged so that bars in the upper row rest in the spaces between the lower bars. All bars are free to move a few millimetres vertically. Since the upper bars are slightly more dense than water and the lower bars are slightly less dense than water, the two sets of bars are pressed together when immersed in water. Ascending mineral-laden bubbles encounter the first barrier where they collect in the spaces between the lower bars. When sufficient bubbles have collected they lift the upper bars and escape upwards against the down flow of wash water. This cleaning action is repeated at each barrier in turn until a mineralised froth, free of entrained gangue particles, forms on the surface of the water. The wash water used is not much more than the quantity of water removed in the froth and yet the use of five or six barriers permits an almost perfect separation. The complete unit is slightly taller than a normal mechanical machine and its capacity is not lower than that of a mechanical machine with the same cross sectional area.

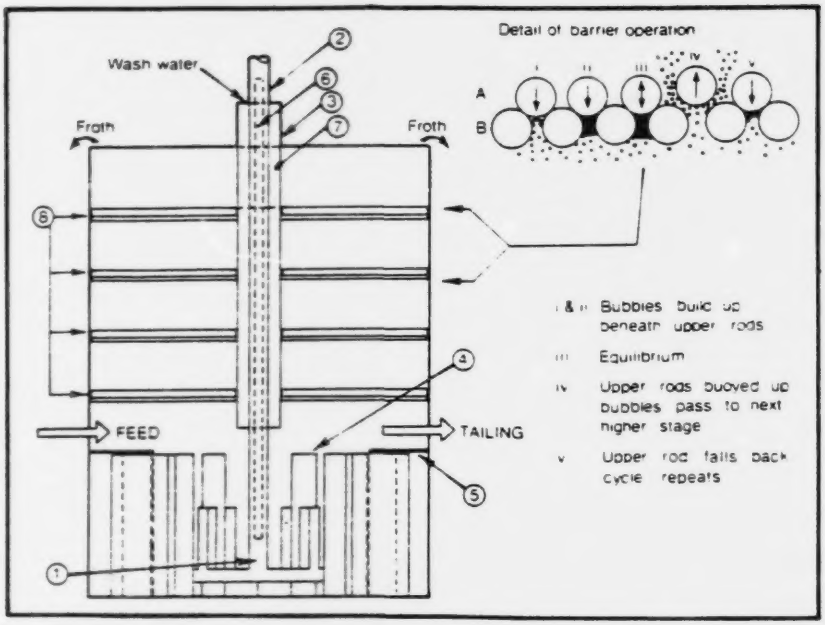

Wash water
Froth
Froth
FEED
TAILING
①
②
③
④
⑤
⑥
⑦
⑧
Detail of barrier operation
i
ii
iii
iv
v
A
B
i & ii Bubbles build up beneath upper rods
iii Equilibrium
iv Upper rods buoyed up bubbles pass to next higher stage
v Upper rod falls back cycle repeats

FIGURE 53: Schematic illustration of a single Norton-Leeds column cell. (1) Impeller; (2) Shaft; (3) Partition box; (4) Baffle vanes; (5) Corner baffle plates; (6) Dual air line; (7) Wash water chamber; (8) Barrier levels - four shown.

A multistage reciprocating plate laboratory flotation column has been developed by Al Taweel and co-workers, Chemical Engineering Department, T.U.N.S.[289,290] The 102 mm diameter column is made from plexiglass. It consists of four stages, partitioned by stainless steel plates with 70 mm centre holes. The agitation is provided by perforated plates mounted on a reciprocating shaft. The perforated plates are positioned near the bottom of each compartment so that intensive agitation takes place in the lower part of each compartment and quiescent conditions exist in the upper parts.

Air is introduced at the base and feed at the middle of the column. The reciprocating plate column generates large bubble population densities. It separates in a counter-current multistage manner and is, therefore, suitable for the recovery of finely ground minerals by flotation. On finer than 75 μm Devco Prince coal, high recoveries of clean coal were obtained. When developed to industrial scale, multistage reciprocating plate columns should be useful for ash and sulphur reduction of coal fines.

2.10.4.4 Froth Separation

Froth separators were developed in the U.S.S.R. in 1961 and by 1972 they had treated 8 Mt of various ores.[266] The principles of operation of a froth separator are shown in Figure 54. The pulp is introduced at the top of the machine and descends over sloping baffles before entering an aeration trough, where it is strongly aerated, and then overflows horizontally onto the top of the froth bed. Particles separate according to their hydrophobicity, less hydrophobic particles pass through the froth while the more hydrophobic are retained and removed by scrapers. This separation method gives ideal hydrodynamic conditions, especially for the separation of coarse particles. Water and solids which penetrate the froth bed pass between aerator pipes into the pyramidal tank. The aerators consist of rubber pipes with 40 to 60 fine holes per cubic centimetre into which air is blown at 115 kPa (17 lb/in^2). The machine has two froth discharge lips, each 1.6 m long and is capable of treating 50t/h solids at 50% to 70% concentration by mass with an air consumption of 2 m^3/min.

The advantages of froth separation are connected with its ability to float coarser particles at very much higher rates (up to ten times) than conventional froth flotation. Thus

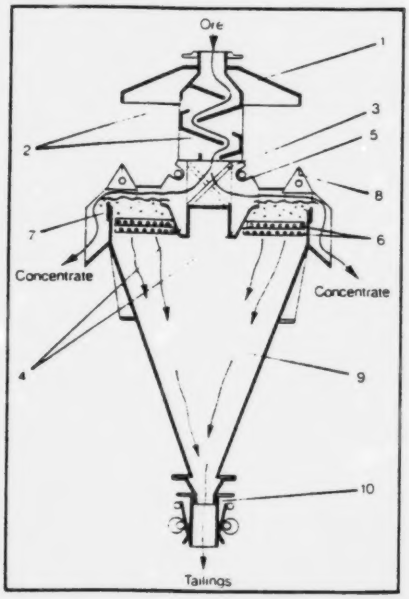


FIGURE 54: Soviet froth separator. (1) Feed tank; (2) Sloping baffles; (3) Divider; (4) Primary aeration channel; (5) Nozzles; (6) Aerators; (7) Overflow lip; (8) Froth paddle; (9) Pyramid-shaped tank; (10) Valve.

they have lower equipment costs and power consumption. The upper size limit of flotation is also increased up to 3 mm. Therefore, for ores which may be liberated at coarser sizes, froth separation will reduce grinding and dewatering costs. The role of flotation time is reversed compared with conventional flotation as increased flotation times give lower recoveries and higher grades. While froth separators extend the upper size limit of flotation, they are not recommended for feeds containing fines. A typical feed size range is stated to be 75 μm to 2 mm and so efficient fines removal is a prerequisite, otherwise the capacity of the machine will be greatly reduced.

Froth separators may be useful for floating coal, possibly between 2 mm and 0.15 mm. These machines appear to have very high throughputs and low equipment and operating costs.

2.10.4.5 Static Flotation Machine

A new flotation technique which is said to improve mineral recovery over a wide particle size range of 3 mm to 5 μm has been developed by the Institute of Physical Chemistry, Bulgarian Academy of Sciences.[291,292,293] The method referred to as "static" flotation is characterised by a long contact period between particles and bubbles and by low inertial and drag forces on particles. The great difference between the predicted maximum size of floatable particles and the actual is caused by inertial and drag forces exerted on the particles in their passage through the pulp, as these forces tear off larger particles from bubbles. In a static system one may realise maximum flotation particle sizes close to the theoretical sizes. The static flotation machine is shown schematically in Figure 55. The conditioned ore pulp is fed from a feed box (1) onto a stationary drum (2) which has a porous surface. The thin layer of pulp descending on the drum surface is exposed to air bubbles emerging through the porous surface. The air pressure required is below 300 kPa (4.5 lb/in^2). The bubbles passing through the few millimetres thick suspension layer permit almost static conditions between particles and bubbles with long contact periods. The froth obtained in this manner consists of an approximate monolayer of bubbles which flows down on both sides of the drum onto the pulp surface. Attached particles overflow into the side chutes (6) while unattached particles sink and are discharged through the lower opening (9) of the trough (3). Depending on the pulp's frothing characteristics, the floated product in the free surface of the pulp may appear as a monolayer of particles or as a mineralised froth. The froth may be broken down on the surface by antifoam devices (8) which may work on ultrasonic,hydraulic or electrostatic principles. Foam breakdown aids selectivity. In a continuous industrial set-up several static flotation machines would be operating in series, the pulp passing from one to the next with a combined float product. Tests carried out with static flotation machines and with conventional Wedag cells on arsenopyrite-cassiterite and felspar-quartz mixtures and also on a lead sulphide ore gave improved performance with the static method.

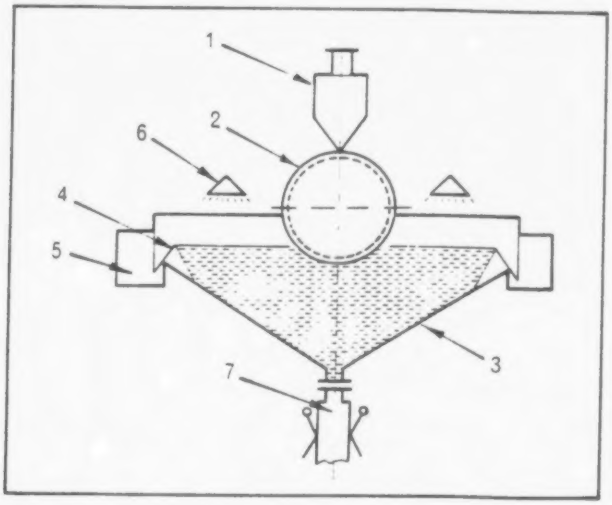


FIGURE 55: Static flotation machine.
(1) feedbox, (2) fixed drum with porous surface, (3) trough, (4) overflow lip (5) chutes, (6) antifoam devices, (7) underflow removal valve.

2.10.4.6 Air-Sparged Cyclone

Preliminary test results for an air sparged hydrocyclone indicate that it gives similar coal separation efficiencies to those of conventional flotation cells.[294] The air sparged hydrocyclone consists of a steel jacket containing a porous cylinder for the dispersion of air. Various configurations of the feed inlet and product outlets have been tested; the preferred arrangement is shown in Figure 56. The fine air bubbles sweep through the conditioned pulp and attach themselves to coal particles. Due to buoyancy, the coal/bubble aggregates are driven toward the cyclone axis and leave through the vortex finder. Non-floating particles move radially outwards and leave through a tangential outlet.

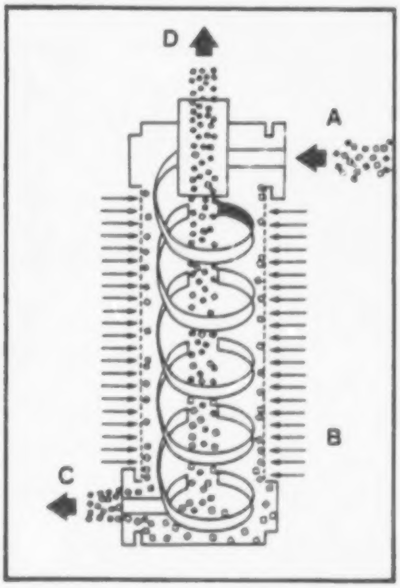


FIGURE 56:

Schematic diagram of the air sparged hydrocyclone. A: Fine particles, requiring high force fields in order to float, are injected at the point of maximal centrifugal force. B: Injection of air through the porous cyclone wall, where the shear rate is maximal, ensures the formation of small air bubbles. C: Hydrophilic particles rejected with the major portion of the water. D: The stabilizing properties of the froth and the low circulation velocities in the centre of the cyclone create a very quiescent froth column for hydrophobic particles.

Since the particles and bubbles are subjected to centrifugal forces, several times those of gravity, the separation is rapid. Retention time is only about two seconds. The generation of a large number of finer bubbles and the directional, rather than random interactions of particles with freshly formed bubbles, improve flotation conditions. The probability of particle-bubble collision and attachment, especially for fine particles, is said to be superior. The 150 mm dia. air sparged cyclone used in the tests on a minus 0.59 mm coal feed, containing 50%-70% finer than 38 μm, gave 0.9t/h capacity with results equalling those from batch flotation. The separation was vastly superior to that obtained with water-only cyclones. Comparison of air sparged cyclones with conventional flotation cells, shows that they have about thirty times higher throughput per unit volume, one-third energy consumption and cost about half.

2.10.4.7 Electroflotation

In electroflotation gas bubbles of oxygen or hydrogen are generated by electrolysis. Conventional flotation cells produce bubbles ranging from 0.6 mm - 2 mm while in electroflotation bubbles are 0.02 mm - 0.1 mm. These small bubbles are ideal for fine particle flotation.[233] Both the rate of bubble evolution and bubble size are determined by the current density. The bubbles in electroflotation are highly active and have a narrow size distribution. In cassiterite flotation tests a current density of 400 A/m^2 was found to be optimum.[295] Although electroflotation appears to be superior for the flotation of fines the specific energy requirements are excessive. In the tests it was found that energy requirements were fifty times greater than for conventional flotation. It should be possible to reduce energy consumption by careful design of the electrode system but it is unlikely that energy requirements will fall below 30 kWh/t, which is still about twenty times those of mechanical flotation machines.

2.10.4.8 Laboratory Flotation Machines

The mechanical agitators of laboratory batch flotation machines are usually scaled down models of agitators used in conventional machines. The new Leeds open-top laboratory cell, designed by Dell of the University of Leeds, departs from this practice.[296] Dell set out to design a laboratory batch flotation cell which is efficient in operation, convenient in use and allows the flotation variables to be set reproducibly. A schematic diagram of this cell is shown in Figure 57. As the name suggests the top of the cell is open which allows easy filling, froth removal and greater flexibility in changing cell geometry. The impeller speed, aeration rate and pulp level are controlled the same way as in the Leeds autofloat cell. The cell body has a capacity of 3 litres and is removable. Extra sections may be fitted on top of the cell body to increase its height. The froth overflows on all four sides of the cell. The cell may be emptied through a drain valve. Low pressure air is piped to the centre of the impeller. The aeration rate is set by a flow control and the rate is indicated on a two stage flow meter. The impeller consists of a disc with upward pointing rods spaced around its periphery. The impeller speed is controlled by a thyristor controlled DC motor which drives the impeller shaft by a toothed belt. The impeller speed is measured on a digital tachometer. A toroidal flow pattern is induced by appropriate vanes incorporated in the cell. The froth may be scraped over the full width of the cell by a notched scraper. This arrangement gives excellent reproducibility. The pulp level is adjusted automatically. A capacitance sensor detects the water level in a manometer tube and admits water to the cell through a solenoid valve when the pulp level drops. Laboratory tests indicate that the reproducibility of the recovery against flotation time relationship of the open-top cell is far superior to that of a conventional mechanical laboratory batch flotation machine. Flotation rates are also higher, resulting in short residence times. Mineral processors, who have attempted to standardise flotation conditions using laboratory batch flotation machines, will appreciate the improvements achieved with the open-top cell in keeping the main variables constant and in obtaining reproducibility.

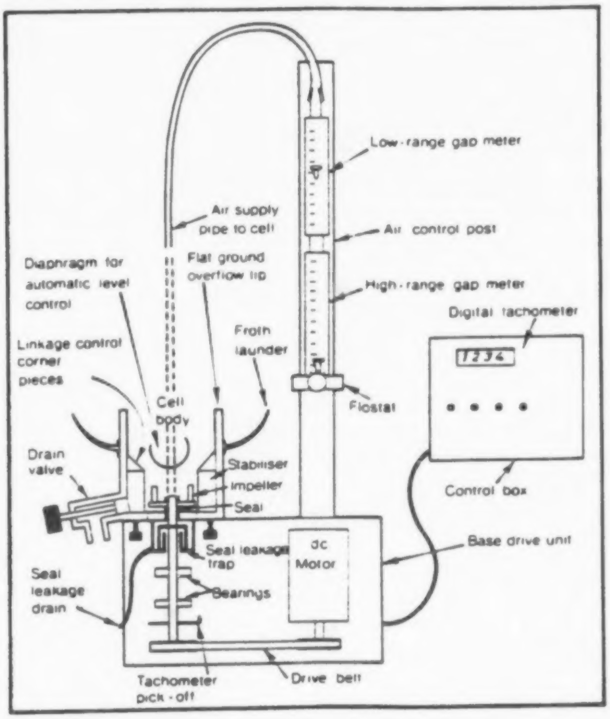

Low-range gap meter
Air supply pipe to cell
Air control post
Diaphragm for automatic level control
Flat ground overflow lip
High-range gap meter
Digital tachometer
1234
Froth launder
Linkage control corner pieces
Cell body
Flostat
Drain valve
Stabiliser
Impeller
Seal
Control box
dc Motor
Seal leakage trap
Base drive unit
Seal leakage drain
Bearings
Tachometer pick-off
Drive belt

FIGURE 57: The Leeds open-top laboratory flotation cell.

Flotation tests should be reproducible and independent of the actions of the operator. With these aims various modifications have been carried out to existing laboratory batch flotation machines. Roberts, Firth and Nicol modified the Denver Model D1 laboratory cell by fixing a plexiglass insert onto the impeller housing stem to improve froth removal.[297] The bottom of the insert is angled at 45° to guide the mineralised bubbles to the front of the cell. With the insert in place, the froth removal may be standardised by using a stainless steel scraper designed to cover the full width and to remove the froth to a fixed depth. This modified cell also uses a constant level control. The ISO standard for coal flotation includes the above modifications.

Another modification was applied to the Denver Model D12 laboratory batch flotation machine.[298] The major features of this improvement include two plexiglass cells, 4 ℓ and 2 ℓ capacities, for rougher and cleaner flotation. The froth removal is mechanised and uses two two-bladed plexiglass paddles on the sides of the cell. The speed of the paddles is adjustable. The pulp level is electronically controlled and the air flow rate is adjustable. Tests indicate good reproducibility of data obtained with modified flotation machine.

2.10.5 Flotation Kinetics and Models

In mineral flotation, much has been published on the interpretation of bench scale data for scale-up purposes or for the evaluation of the performance of flotation plants. The flotation process is often described by a first order rate equation, whose rate constant is determined from test data. Flotation kinetics are detailed in many references and no attempt has been made here to summarise this broad topic.[299,300]

The selection of flotation time which gives optimum performance is derived by Agar and co-workers.[301] They found that the criteria for optimum flotation time may be stated in three different ways:

1. Accept no material to the concentrate that is lower grade than the feed to that separation stage.

2. Maximise the difference in recovery between the useful mineral and the gangue.

3. Maximise the separation efficiency.

Since these three statements express the same condition in three different ways, they all give the same optimum residence time. Agar has also shown how laboratory batch data may be used for the prediction of continuous flotation[302,303] and for reagent optimization.[304]

Flotation models are useful in machine design for incorporation into automatic control and for diagnosis of flotation performance.[305] Mathematical models may be used for evaluating the effects of flotation variables on the residence time of feed solids.[306] As a result of process modelling of both pilot and production scale plants, Cutting, Barber and Watson report that flotation machines in current use are not designed for efficient froth removal.[307] More efficient means of froth removal from the flotation cell are needed to reduce collapse and recycling of material at the rear of the cell. The use of identical flotation cells for roughing, scavenging and cleaning is unsatisfactory as in the cleaner and scavenger cells froth removal from the rear is particularly poor. The overall efficiency of flotation circuits could be improved by separating the rougher concentrate into at least two particle size ranges before cleaning. This should prevent recycling of coarser material between flotation stages and reduce circulating loads. This scheme may also reduce the need for extensive scavenging circuits in the mill.

2.10.6 Flotation Circuits

In mineral processing, elaborate flotation circuits are often used, incorporating rougher, scavenger, cleaner stages, frequently with regrinding within the circuit. Sometimes two or three cleaning stages are needed to obtain concentrates with sufficiently high grades. In coal preparation, simple flotation circuits are used as the coal feed is already relatively high grade. Nevertheless, the same principles apply as to the more complicated mineral flotation circuits.

Detailed analysis of mineral and coal flotation circuits is given in a recently published book.[308] The behaviour of coal flotation circuits in relation to reagent additions and feed size distributions is discussed. Modelling, simulation and automatic control of circuits are covered in depth.

The performance of a flotation circuit is determined by many factors which influence the process in complex ways. To obtain a complete model of the process each factor must be accounted for in the equations. The factors may be determined by:

1. feed, reagents and process steps.

2. flotation machine design and operating variables.

3. pulp transport characteristics.[305] The mathematical model should be accurate and at the same time simple enough for easy computer evaluation. Numerous models have been developed for sulphide flotation circuits.[309-311]

Firth, Swanson and Nicol discussed the selection of flotation circuits for poorly floating Australian coals.[241,312] They found that the feed size distribution was too wide for adequate flotation and the reagent distribution was unsatisfactory. Improvement in flotation performance could be achieved by split conditioning, two stage reagent addition and by reflotation of classified tailings.

2.11 Selective Flocculation

Selective flocculation is possible using ordinary natural or synthetic polymer flocculants provided that the electrokinetic potentials of minerals are carefully controlled. There have been many attempts to increase the selectivity of the process. It is possible to make the polymer preferentially adsorb on a metalliferous mineral by using synthetic polymer flocculants containing chelating groups which are specific for a certain metal.[313] In another method a flotation collector is adsorbed on the useful mineral and the polymer flocculant preferentially adsorbs on the treated surfaces by hydrophobic association.[314]

Selective flocculation is used industrially in mineral processing. The Cleveland Cliffs Iron Company, Michigan, started operation in 1974 treating 29000t of iron ore daily by selective flocculation followed by flotation.[315,316] Here the finely ground ore, 80% finer than 32 μm, is treated with a starch flocculant. The hematite is selectively flocculated and sediments in a thickener while the silicates remain dispersed and overflow. The preconcentrate is further upgraded by selective flotation using an amine collector while the hematite is depressed.

The feasibility of recovering coal by selective flocculation has been examined independently by several investigators, but the process has not been used industrially. Selective flocculation of shale by the sodium salt of carboxymethylcellulose has been tested.[317] In this method the coal remained in suspension and the shale flocculated and the clean coal produced was reported to be of low ash content.

The U.S. Bureau of Mines has conducted laboratory scale investigations of selective flocculation applied to minus 38 μm coal and shale.[318] They found that ultra-fine coal particles may be selectively flocculated in the presence of associated mineral matter. The quantity of dispersant needed to stabilise the suspension was 1.5 to 4.0 kg/tonne. Low molecular weight non-ionic synthetic flocculants at 0.05 to 0.25 kg/tonne functioned selectively and alkaline conditions (around pH 11) were found to be favourable. The flocculated sediment remained a high ash product due to entrapment of shale and required further treatment. It was concluded that selective flocculation of coal would be prohibitively expensive, would not produce a low ash coal product and has no immediate future in the coal industry.

Brookes and co-workers of Nottingham University, England, have obtained separation with a reagent, a polymer containing a creosol component, specially prepared by Allied Colloids Ltd.[319] However, when compared to flotation, selective flocculation is unable to compete for separation efficiency. Solari and Rubio in Brazil, using natural and synthetic flocculants also found that selective flocculation did not give sufficient selectivity in coal/shale separation.[320] Attia and Fuerstenau, on the other hand, found that coal could be selectively flocculated with Purifloc A-22, a high molecular weight polymer flocculant, in the pH range of 9-11.[321] But they found that dispersion of shale particles before flocculation needs special attention. They also managed to selectively flocculate pyrite with a poly-xanthate dispersant and to separate the flocs from dispersed coal and shale. Thus, it appears that in selective flocculation the entrapment of unflocculated material is a serious problem. At Cleveland Cliffs, selective flocculation is a preconcentration process followed by flotation for the removal of entrapped material. This separation sequence may not be applicable to coal. In mineral processing the redispersion and reflocculation of the selectively flocculated product have been successfully tested, but the process became complex and reagent costs were high. Another disadvantage of selective flocculation results from the open structure of flocs which increases the moisture content of the product.

2.12 Oil Agglomeration

Selective agglomeration involves agitation of the coal/shale slurry in the presence of an oil. The oil droplets adhere to the naturally hydrophobic, high rank coal particles which agglomerate selectively leaving the shale dispersed in the water. The coal agglomerates may be separated from the dispersed shale simply by passing the agglomerated slurry over a screen. Pilot scale tests have indicated that selective agglomeration gives lower ash content and lower moisture content of the clean coal product than froth flotation followed by filtration.[322,323] Considerable savings could be realised as the process requires relatively simple equipment and works at higher pulp densities. Oil agglomeration is advantageous when: 1. The proportions of very fine coal are high. 2. The coal is oxidised or contaminated with fine clay and therefore the flotation yield would not be high and the clean coal ash sufficiently low. 3. The cost of oil used in the process is not lost. Often the oil in agglomerated coal is regarded as moisture because most of it evaporates in moisture content determination. 4. Costs of dewatering and drying are high. 5. A free flowing and dust-free product is desired.[324] Oil agglomeration was used in four plants in 1926, even before flotation became an important process. In the United States, the largest of the four plants produced 600 t of 25 mm agglomerates daily, containing 30% fuel oil by weight. However, the price of the agglomerates was high and the plants closed after 6 to 8 years. The next major step in oil agglomeration was the German Convertol process investigated and tested between 1952 and 1965 to reclaim fine metallurgical coal for coking. Due to the cost of oil the Convertol process was too expensive to be competitive.

A detailed review of the development of oil agglomeration is given by Mehrotra, Sastry and Morey.[325] The first large scale industrial use dates from 1922, in Alexandria, Virginia, when it was known as the Trent process. More recent work on oil agglomeration includes the National Research Council of Canada, (NRCC), process,[326-329] the Shell Pelletizing Separator, (SPS), process, Netherlands,[330-331] the Olifloc process of the Bergbau Forschung GmbH, Germany,[332,333] the Central Fuel Research Institute, (CFRI), Dhanbad, India[334] and the Broken Hill Proprietory, (BHP), process, Australia,[335-337] (Table 1).[325]

Apart from ash removal, oil agglomeration is also useful in pyritic sulphur reduction.[338,339] Pyritic sulphur reductions of up to 70% have been reported. An interesting way of making pyrite removal more selective is by the use of Thiobacillus ferrooxidans which renders the pyrite hydrophilic by bioadsorption.[340] The same principle could be applied to the flotation of coal with improved pyrite depression.

TABLE 1

Extent of development of major oil agglomeration processes

Organization/Process	Laboratory investigations	Pilot plant investigations	Commercial plants
Trent	Tests on nineteen different coal/refuse samples. Removal of sulfur by preparing a synthetic mixture of pyrite and coal (Perrott et al., 1921). Comparison with froth flotation. Influence of type of oil, and fines on recovery and ash content (Ralston, 1922). Influence of oleic acid on recovery (Gaudin, 1932)	None reported	600 t/d at Alexandria, Va (1922); Four other U.S. plants, two installed outside U.S. Not currently practiced
Convertol	Influence of oil type, sp. gravity, amount, pulp temp, pulp density, retention time, agitator speed on combustible matter recovery (Sun and McMorris III, 1959)	Pilot plant at U.S. Steel Corp., Gary, W. Va (1955). Plant capacity 3—5 t/h fine coal from slurry containing 35—45% solids. Another similar plant at U.S. Steel Corp., Greensboro, Pa (Brisse and McMorris, 1958)	Bergwerksverband zur Verwertung von Schutzrechten der Kohlentechnik, GmbH., Germany (1952—1965)
NRCC (spherical aglommeration)	Effect of oil amount on agglomeration. Effect of suspending liquid density and amount, feed size distribution, agitator speed, pulp density and slurry pH. Oil type on agglomeration (Capes et al., 1961, 1970, 1971, 1976)	Pilot plant in West Virginia with capacity of 0.5 t/h of product. Effect of operating variables on ash content and recovery at two similar pilot plants in Eastern Canada	Not currently practiced. None reported
Shell	Response of SPS to various types of oils. Power consumption in operating SPS (Zuiderweg and van Lookeren, 1968; van der Meer and Quist, 1972)	Comparison of pilot plant SPS (capacity 10—20 m^3/h of soot suspension) with laboratory SPS (Zuiderweg and van Lookeren, 1968). 25 t/h plant for dewatering of fine coal at Fryston, U.K.	10 t/h at Mitsui Mining, Japan (1972—1975)
Olifloc	Effect of operating conditions such as pulp density, quantity of fuel oil, oil consumption as a function of type of oil (Bogenschneider and Jasulaitis, 1977). Loss of oil with waste water stream	Pilot plant of Monopol Coal preparation plant of Ruhrkohle AG with maximum throughout of 3 t/h solids. Investigations on the behavior of flotation and filtration circuits after incorporating oil agglomeration in the process flow sheet	25 t/h at Prosper III, Germany, other plant construction at Minister Stein preparation plant
CFRI	Effect of conditioning time, stirrer speed, pulp density, types of oils, oil characteristics and pH on agglomeration behavior of several types of coals (Sarkar et al., 1979)	Pilot plant at Lodna (India) set up by Bharat Coking Coal, Ltd. capacity 2 t/h. Influence of operating variables and oil types (Sarkar et al., 1979) deoiling characteristics of the product and coking tests	No commercial installations reported
BHP	Effect of pre-emulsification, oil type and amount of oil, comparison with froth flotation (Nicol and Swanson, 1979; Bensley et al., 1977). Influence of presence of clay minerals, stirrer speed, feed size distribution (Swanson et al., 1977a, b)	Pilot plant at BHP facility in Australia — treats 3 t/h of reject slurry. Settling behavior of tailing before and after oil agglomeration, and coking tests (Nicol and Swanson, 1979)	5 t/h — Semi-commercial plant at BHP facility in Australia (1978) (Nicol and Swanson, 1979)

The advantages of oil agglomeration compared with flotation or selective flocculation are:[341]

1. High coal recovery attained.

2. Easy dewatering of fine coal since compact agglomerates are formed.

3. Recovery of coal from fine waste slurries simplifies fine refuse treatment, by increasing the settling ratio of the lower solids concentration refuse slurry. Tailings pond life is also extended.

Application of mineral processing techniques may enhance the selectivity of oil agglomeration. More effective use of flotation collector type surface active agents for coal should be investigated.

2.13 Coal-Liquid Mixtures

The idea of coal-in-oil mixtures (COM) as a useful alternative fuel has been explored at intervals for over one hundred years.[342] To conserve oil resources the U.S. Department of Energy vigorously promotes the idea. Fine coal mixed with oil may be burnt similarly to oil, but the mixture has to exhibit sufficient stability to prevent settling out. For combustion, grinding to 80% passing 75 μm is adequate but this fineness does not give the mixture sufficient stability. Inducing stability by fine grinding results in excessively high mixture viscosity for pumping and combustion. Grinding and dispersant addition have proved more successful. Gellants such as heavy metal soaps, thickeners and fillers have also been tried. The addition of water, which acts as a flocculating agent, and in very fine droplet form thickens the oil phase, is effective but 3% to 8% is needed. Another type of liquid fuel is coal-water fuel (CWF), in which fine coal particles are dispersed in water.[343] A low ash content coal is preferable for CWF as boilers designed for oil firing would convert directly to accept 1% to 2% ash but with modification would accept 3% to 6% ash. Since CWF contains about 30% water, which evaporates during combustion, the fuel uses about 3-5% of its own energy for this evaporation. Fine grinding of coal and surfactant addition make CWF sufficiently stable. A wide variety of surface active agents has been tested as additives to promote stability. The problem of freezing may be overcome by the addition of antifreeze compounds. Alcohol type antifreeze additives also improve combustion characteristics.

No attempt is made here to summarise coal-liquid mixtures technology. Detailed discussion may be found in the Proceedings of the Second Annual Coal-Liquid Mixtures Workshop, Halifax.[334] The Third Annual Coal-Liquid Mixtures Workshop, Halifax was held in October 1983 but the Proceedings are not yet available.

The application of mineral processing technology to coal-liquid mixtures has been given in the discussion of the relevent unit operations.

2.14 Pyritic Sulphur Removal

Since organic sulphur is fairly uniformly distributed in coal, physical separation methods are restructed to pyritic sulphur removal. But, where the coal constituents vary in organic sulphur content, if a selective collector could be found to differentiate between high and low sulphur coal, then separation should be able to produce a lower sulphur content product, e.g. for coking, and a higher sulphur content product for power station use. There is abundant literature on sulphur reduction. In the U.S.A., the use of coal is being expanded, but at the same time, the Environmental Protection Agency's New Source Performance Standards (NSPS) require up to 90% SO_2 removal from power station flue gases.[345] Thus, to comply with the limit of 1.2 lb of SO_2 per million Btu heat input, coal desulphurization became the focus of attention and much research has been carried out and the findings have been published. Books appeared dealing with the topic[346-348] and other books written on coal cleaning contain sections on desulphurization.[349-353] Symposium proceedings also contain much information.[354]

Low density separation in the region of 1.3 g/cm^3 appears to give good sulphur reductions for Cape Breton coals.[355] Clyburn discusses current and new technologies applicable to the desulphurization of DEVCO coals, including reverse flotation, Carbogel process and coal-water slurry.[356]

Several comparative studies have been published on sulphur reduction by flotation and high gradient magnetic separation.[352,353] These comparisons may indicate some expectations of performance obtainable from the different processes but any particular coal should be individually tested as pyritic sulphur reduction characteristics may vary considerably. The choice of the most suitable sulphur reduction process for a given coal also depends on the cost of processing apart from the technical performance. When actual processing costs are not available, estimates may vary considerably.

In the earlier sections, the suitability of various techniques to pyritic sulphur reduction was mentioned and will not be repeated here. The following generalisations may be relevent. Physical sulphur removal can never be an ideal separation process since pyrite is only fractionally liberated in the feed. Even fine grinding will not give a high degree of liberation. But physical separation is relatively inexpensive and should be used whereever it can give acceptable results. Application of a physical process before further chemical treatment should also be considered. For improved sulphur separation the feed should be treated in narrower size ranges, each size range being treated by the most suitable process. Unfortunately, in flotation. simultaneous removal of shale and pyrite has so far not been satisfactory. Flotation of coal, the greater portion of the feed, inevitably carries shale and pyrite into the clean coal by entrainment. Better selectivity would be achieved if the pyrite and shale could be floated, but shale flotation may not be feasible. Two stage flotation, with conventional flotation in the first stage, followed by depression of coal and pyrite flotation, appears to give better results but this process is more complex and expensive. The choice of sulphur reduction method is most likely to be governed by the characteristics of the coal and by market requirements for the clean coal products.

2.15 Separation by Coal Petrography

The petrographic composition of coal is industrially important as it determines the coking and carbonization properties. Petrographic composition has been used to predict the strength of blast furnace coke.[357] Knowledge of the desired petrographic composition of a clean coal product, depending on its intended use, may make it possible to enrich the required petrographic components by means of suitable separation techniques. In laboratory tests on a high volatile bituminous coal it was shown that the density of exinite was below 1.25 g/cm^3, of vitrinite was between 1.25 - 1.35 g/cm^3 and of inertinite was above 1.35g/cm^3.[358] Thus, in dense medium separation of fines it should be possible to enrich the desired component.[359] Other physical properties such as differences in breakage characteristics and in electrical properties may also be utilized. Differences in surface properties could be exploited in flotation, oil agglomeration and selective flocculation. Cook outlines the petrology of Australian coals.[360] He gives a more precise categorization of mineral matter in coal and points out that coal is heterogenous at different levels. Even a certain maceral is not of uniform chemical composition. Other heterogeneties are compositional variations between macerals, banded components and seams. This inhomogenous nature makes it possible to separate coal at different levels, by removing coarse shale, finer shale and pyrite or by separating petrographic components.

2.16 Dispersion, Coagulation and Flocculation

The stage of a suspension is a determining factor in the separation process. Initially, the feed to any mineral separator should be dispersed so that individual particles may be separated according to their mineral contents. For solid-liquid separation, a coagulated or a flocculated suspension is preferable, since the flocs have greater sedimentation velocities in thickening and lower specific resistances in filtration than individual particles. Dispersion utilizes electrical repulsive forces between particles to keep them apart. Most frequently, inorganic reagents are used as dispersants, but low to medium molecular weight synthetic polyelectrolytes are also used. Coagulation of particles occurs when the electrokinetic potentials are reduced and the double layer is compressed by the addition of an electrolyte so that the relatively short range van der Waals attractive forces are able to hold particles together. Coagulation also occurs when particles of differing signs of charge are present in a suspension. Most minerals are negatively charged around neutral pH, but some, such as alumina, are positively charged. Higher valency cations cause coagulation at low dosages. The most commonly used cations for coagulation are Ca, Fe and Al. Coagulation is a reversible process as, after disruption, flocs reform. Coagulated flocs are small and feeble. Flocculation is achieved by adsorption of a natural or synthetic polymer by solid particles. The main mechanism of adsorption is hydrogen bonding. High molecular weight synthetic linear polymer flocculants are available with molecular weights up to 25 million. These polymers are able to bridge between particles and adsorb onto two particles, thus holding them together. Since flocculation is a random process of particle aggregation, flocs have an open structure. Flocs resulting from the use of synthetic polymers are large and compressible, but once disrupted are unlikely to reform. Flocculated suspensions have much higher sedimentation velocities than dispersed or coagulated suspensions.

This is an advantage in solid-liquid separation. Thus flocculation is useful in thickening, centrifuging and filtration. The introduction of synthetic polymer flocculants has greatly aided solid-liquid separation of fine suspensions.

The theoretical aspects of flocculation are covered in "The Scientific Basis of Flocculation," edited by Ives.[361] Moss reviews the mechanism of and factors affecting flocculation.[362] Flocculation proceeds in two stages: the adsorption of reagent and the formation of aggregates. Adsorption of flocculant molecules may take place by hydrogen bonding (the most important method), by formation of insoluble compounds at the surface of particles by the flocculant molecules or by electrical attraction. Individual hydrogen bonds are relatively weak. However, on a high molecular weight long chain flocculant molecule several thousand hydrogen bonds are available and represent a high overall bonding force. Flocs held together by synthetic polymer flocculants can be degraded by excessive agitation and cannot normally regenerate since polymer molecules tend to be adsorbed on the same particle. Therefore flocculated pulps should not be sheared excessively prior to thickening, filtration or centrifuging. The pulp pH modifies surface charges on particles and changes the shape of flocculant molecules by either coiling or extending them. The change in molecular configuration in turn affects flocculation efficiency because coiled or rigidly extended molecules are not as efficient as flexibly extended ones. When selecting flocculants the operating pH range for each reagent must be considered. Thorough but gentle agitation is vital during and after flocculant addition to distribute flocculant and to aid floc formation. High local flocculant concentration due to poor mixing is wasteful. For best performance flocculants should be added in dilute solutions by multipoint addition.

The practical aspects of the use of flocculants and surfactants for enhancing dewatering in mineral processing are discussed by Hunter and Pearse.[363] Synthetic flocculants are long-chain, predominantly linear, water soluable polymers which can adsorb onto particles in a suspension causing particle aggregation. Flocculants may be divided into two main classes. The first group includes medium to high molecular weight (10^6 to 20×10^6) bridging flocculants, based on polyacrylamide and available in non-ionic, anionic and cationic forms. The non-ionic reagent is usually polyacrylamide. Anionic forms are copolymers of acrylamide and sodium acrylate. The cationic forms are copolymers of acrylamide and a cationic function group. In both anionic and cationic types the relative proportions of the copolymers may be varied to give flocculants ranging from non-ionic to fully anionic or cationic in character. Thus a flocculant can be tailor-made to suit virtually any pulp characteristics. The second class of flocculants is of much lower molecular weight (less than 0.5×10^6) and is highly cationic. These flocculants are usually supplied as 10% to 50% solutions. There are two main types in the second class, polyamine quaternaries, which are used as clarification aids in coal preparation and poly (diallyldimethylammonium chloride) uses extensively for water clarification in iron ore concentrators. A wide variety of flocculants is available from manufacturers. For a certain specified slurry the flocculant selected should be the most economical one which produces the required float characteristics. As a broad generalisation, their areas of application may be summarised as follows: 1) Non-ionic and anionic polymers of high molecular weight are most frequently used in mineral processing. 2) Non-ionic flocculants are widely used in acidic pulps. 3) Highly anionic flocculants are suitable for alkaline pulps. 4) Medium molecular weight polymers are more suitable for filtration and those with high molecular weights for thickening. The appropriate dosage levels of flocculants span a wide range. Some slurries may be adequately flocculated by 0.003 to 0.025 kg/t flocculant while others may need 0.025 to 0.1 kg/t, or an even higher dosage to obtain a strong sediment. The paper outlined flocculation mechanism and the main factors which affect the process. It explains laboratory evaluation of flocculants, describes preparation and applications of flocculants in plants and covers their economic aspects. This paper contains much useful information on the industrial use of flocculants and surfactants.

The selection of the most suitable flocculant for a certain duty is carried out by laboratory batch flocculation and sedimentatin tests in a graduated cylinder. At various flocculant dosages, the sedimentation rate and supernatant liquid clarity are evaluated. In these tests standardisation of procedures is most important for meaningful results. International standards (ISO) for flocculation testing are in preparation. The most suitable flocculant is the one giving the required performance at least cost. Hamza has developed a graphical method for the determination of optimum flocculant dosage which gives the minimum total cost of solid-liquid separation.[364]

An experimental study of the settling characteristics of washery effluent derived from froth flotation tailings was undertaken at Western Research Laboratory, Edmonton, Canada, in order to select the most appropriate flocculant.[365] The 62.5% minus 30 µm slimes were obtained from the cyclone overflow of the flotation tailings at the Cardinal River washery. The overflow contained only 0.6% solids by weight. Settling rates determined in 100 ml stoppered graduated cylinders were used in the evaluation of flocculation performance. A "cost performance index" (CPI), was defined as the cost of flocculant to give maximum settling rate per tonne of solids divided by the settling rate. For this effluent Separan MG 700 gave the highest CPI at an optimum dosage of 0.05 kg/tonne, from a large number of flocculants tested. Different combinations of reagents were also used. Superfloc 330 flocculant aid with various polyacrylamide flocculants was found to have no advantages.

When the use of a polymer does not give sufficiently clear thickener overflow the addition of a second, lower molecular weight polymer, often produces clean water for re-use.[366] The two polymers may be mixed in the required proportions as powders, before dissolution, but care must be taken that the flocculants are compatible. The correct preparation of flocculant is very important. The proper dispersion of the powder in water may be achieved with dispersers such as the Bretby Autex or Allied Colloids Jet-wet dispersers. The higher molecular weight polymers may take two to three hours to dissolve. For best results the polymer should be dissolved in clean, preferably soft, water.

The Auto Jetwet is an improved apparatus for the complete dispersion and dissolution of powder and bead type flocculants in water, introduced by Allied Colloids plc., Bradford, England.[367] The flocculant is fed into the Auto Jetwet by a high volocity air stream and is wetted by intense water sprays. The machine dispenses and dissolves flocculants without the formation of undissolved lumps and it incorporates a self-cleaning action. The standard system can dissolve up to 20 kg flocculant an hour. Allied Colloids also offers a range of flocculant metering pumps equipped with in-line mixing and dilution facilities.

The dissolved and diluted solution may be distributed by metering pumps or by less expensive, constant head flocculant ring mains.[366] The flocculant is often metered in proportion to the solids feed rate. An automatic system for the control of flocculation, called the Clarometer, was developed by the Mining Research and Development Establishment of the National Coal Board, U.K.. The Clarometer measures, at regular intervals, the sedimentation velocity of a pulp sample taken from the thickener and modifies the flocculant dosage rate as required.[368] The system automatically shuts off the flocculant flow when no solids are entering the thickener.

2.17 Thickening

An extensive review of gravity thickening theories has been provided by Warren Spring Laboratory, Department of Industry, Stevenage.[369] The article traces the development of the scientific principles of thickening starting with Mishler's theory, 1912, to Adorjan's theory, 1975, including sixteen different methods for the calculation of thickener area requirements. The author compares the various theories and discusses their limitations. The two theories most generally used by the minerals industry are those of Coe and Clevenger and Talmage and Fitch. The former tends to underestimate thickener area requirments while the latter overestimates. Since a conservative estimate is more acceptable the Talmage and Fitch method is often preferred.

Professor Schubert of the Bergakademie, Freiburg, G.D.R., gave a detailed scientific review of the present state of sedimentation theory and of the sizing of thickeners.[370] He explained the interpretation of batch sedimentation test results and how solids flux plots may be used to determine thickener area requirements. The article also discusses more recent approaches to thickening in which sediment compression processes are considered. In another paper, Wilhelm and Naide advocate the use of a single batch settling test for sizing thickeners.[371] They found good agreement between their predictions for several types of slurries and continuous thickener operating data.

A new method has been developed for the determination of area and height requirements of continuous thickeners.[372, 373] The method assumes equilibrium of compressive, fluid drag and gravitational forces within the pulp and relies on empirical permeability and compression data. This work demonstrates the importance of pulp height in the thickening of flocculated pulps. Thickener dimensions determined from compression and permeability data are compared with those obtained from batch sedimentation tests. Batch sedimentation tests do not provide sufficient information for the design of thickeners for operation of flocculated feeds in which compressive strength is an important factor. The Coe and Clevenger and Talmage and Fitch methods tend to give thickener areas greatly in excess of those required. Other summaries of thickening theories are given by Fitch,[374] Wakeman[375] and Keane.[376] Keane also describes thickening equipment and practice together the economic considerations.

Adorjan outlined some recent developments in thickening.[377] The most important advance in this area has been the industrial acceptance of high capacity thickeners, which usually incorporate some or all of the following features:

1. Mechanical mixing of flocculant and slurry.
2. Stage addition of flocculant.
3. Introduction of flocculated slurry below pulp level.
4. Inclined plates.
5. Sufficient sediment depth to provide reasonable underflow concentrations.
6. Higher than normal raking torque to handle the increased loading.
7. Reliable instrumentation and control.

High capacity thickeners have shorter retention times and operate closer to the limiting conditions than conventional thickeners and so automatic control to optimise their operation and to dampen fluctuations is most desirable.

High capacity thickeners have gained acceptance both in mineral processing and coal preparation plants world wide.[378-381] Today most manufacturers produce high capacity thickeners. These machines are characterised by relatively low thickening areas for a given throughput of 0.03 to 0.06 m^2/tonne/day (0.3 to 0.6 ft^2/tonne/day) compared to 0.5 to 1 m^2/tonne/day (5 to 10 ft^2/tonne/day) of conventional thickeners[376,382] The first of this type was probably the Amstar Corp.'s "Enviro-Clear" thickener. Slurry entering this machine is deflected downwards by the feed well to an impingement plate where horizontal velocity is induced so that the incoming particulate material meets the sludge bed and entrapment of the solid particles occurs. The agitated sludge whose height is maintained by level detectors behaves as a filter. Enviro-Clear units may be equipped with radial as well as peripheral overflow launders. The Eimco-BSP Division of Environmental Corp. offers a "Hi-Capacity" thickener. In this machine the feed enters through a hollow drive shaft where flocculation takes place. The flocculated feed is directed into a slurry blanket. Radially mounted inclined plates improve the thickening operation by creating a flow pattern in which the thickened sediment slides down over the plates and the water released escapes underneath. The slurry blanket height is automatically maintained. Dorr-Oliver has introduced the "Dynafloc" thickener. This design makes use of idealised flocculation conditions, i.e. effective mixing of flocculant with pulp and minimisation of shear forces after flocculation.

The principle of operation of lamella thickeners is based on a nest of inclined parallel plates, situated in a sedimentation chamber, which reduces settling distance and increases effective area.[383] A flow pattern is induced between plates in which clear liquid flows upwards and sediment slides downwards. There are two main types of lamella thickeners depending on the configuration of plates, the counter current and the cross flow and their principles of operation together with a comparison of their performances is given by Schlitter.[384] Cross flow type lamella thickeners are made by Chemie und Metall GmbH, Rheax, Austria.[385] The thickener body is a completely welded rectangular box with a settling trough and settling pocket forming the unit. The lamella packs are removable and a number of plate distances are available. The performance of Lamella thickeners in coal preparation plants has been assessed by the White Ridge Coal Company, West Virginia.[386]

An interesting method, which increases the capacity of a conventional thickener by the installation of Lamella modules, is offered by Sala International of Sweden, (Figure 58). This unit has great potential for use where thickener capacity is limited by slowly sedimenting particles or flocs. It is particularly suitable for plants operating with overloaded clarifying or thickening circuits where incorporation of Lamella modules will increase capacities of existing thickeners. Units are available for thickeners with diameters between 7.5 m and 20 m.

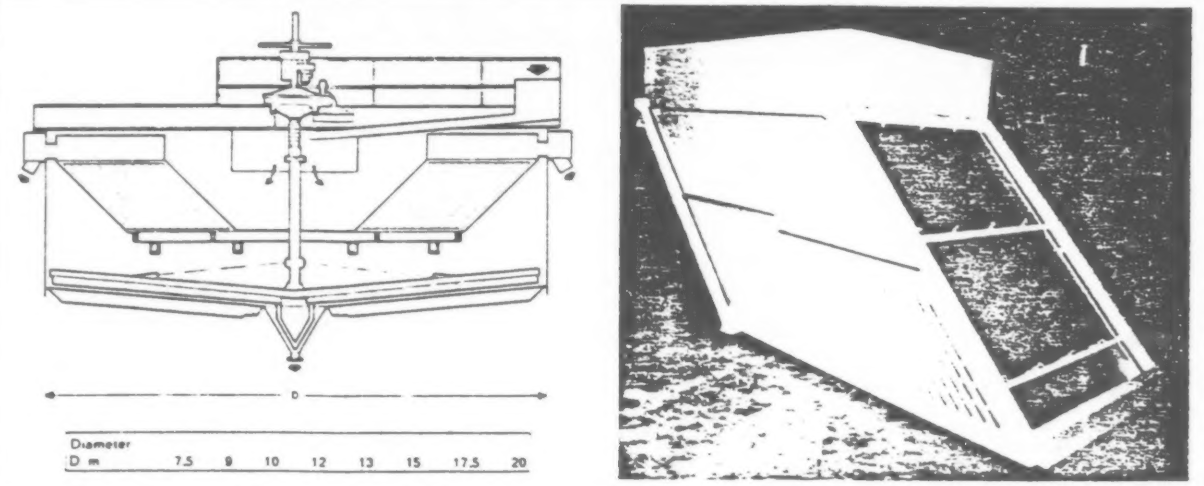


FIGURE 58: Lamella modules (right) from Sala International for incorporation in existing thickeners.

Deep cone thickeners were developed by the Mining Research and Development Establishment of the National Coal Board, U.K. and Allied Colloids Limited about fifteen years ago. They are 4 m diameter conical containers equipped with stirrers and overflow and underflow discharge arrangements,[387] (Figure 59). The thickened pulp discharge is automatically controlled by a pressure transducer sensing the pulp depth in the cone. These thickeners are specially suitable for treating flocculated pulps which require a substantial solids pressure for their compression. The discharge pulp is of such a consistency that it may be transported by belt conveyor. Deep cones have low capital and running costs and their capacities reach 120 m^3 pulp/h. Recently a licensing agreement has been made between the N.C.B. and Allied Colloids Ltd. with Joy Manufacturing Co., Pittsburgh, to manufacture the thickener.

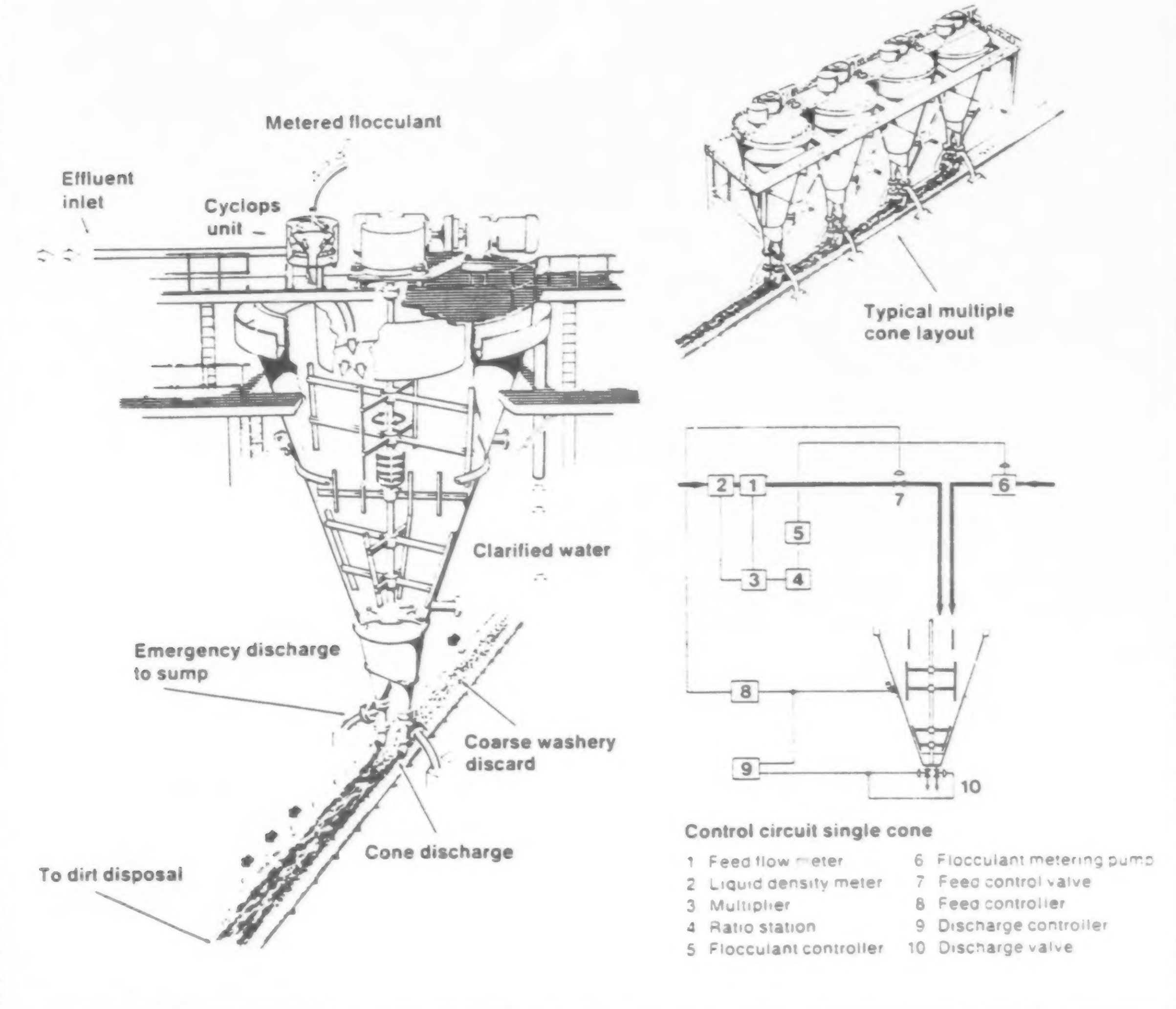

Metered flocculant
Effluent inlet
Cyclops unit
Clarified water
Emergency discharge to sump
Coarse washery discard
Cone discharge
To dirt disposal
Typical multiple cone layout
2
1
7
6
5
3
4
8
9
10
Control circuit single cone
1 Feed flow meter
2 Liquid density meter
3 Multiplier
4 Ratio station
5 Flocculant controller
6 Flocculant metering pump
7 Feed control valve
8 Feed controller
9 Discharge controller
10 Discharge valve

FIGURE 59: Layout of the deep cone thickener.

Experiments carried out in a 600 mm continuous thickener, equipped with electrodes connected to a D.C. supply indicate that gravitational settling of fine suspensions may be assisted by electrophoresis.[388] The upper electrode was the cathode and the lower was the anode so that the electrophoretic force was added to the gravitational force. A minus 400 mesh silica suspension was used in the tests and the thickener was operated sometimes with electrical input to the electrodes and sometimes without. With electrophoretic assistance the solids concentration of overflow was considerably reduced and the recovery of solids in the underflow increased. The overflow clarity increased with increasing voltage gradient and also with lower suspension solids concentration. The electrophoretic mobility of particles was reduced and power requirements increased at higher ion concentrations of the aqueous phase. These tests have demonstrated that electrophoretic forces may be used effectively in the thickening of very fine suspensions, although at present electrophoretic clarification appears impractical technically and economically.

2.18 Dewatering

The simplest form of dewatering is over a stationary or vibrating screen. The lower limit of feed size which is industrially dewatered on screens is about 0.3 mm. However, when oil agglomeration is used, very fine particles may be dewatered in the agglomerates. This emphasizes the advantage of oil agglomeration as a combined coal cleaning and dewatering process.[389]

Centrifugal dewatering of clean coal coarser than 0.5 mm has long been practised. Basket type centrifuges are suitable for this work.[390] Solid bowl centrifuges are able to remove much finer material and they have been used for dewatering fine tailings before disposal.[391] Tailings dewatering is required for the return of process water in those areas where the topography does not permit the use of tailings dams.

Screen bowl centrifuges have been successfully employed for the dewatering of coal fines in Canada, U.S., U.K., Germany and South Africa. In Canada, Fording Coal, B.C. installed two Bird screen bowl centrifuges.[392]
In a search for the most favourable dewatering process for froth floated fines the MRDE of the NCB carried out tests with a screen bowl centrifuge.[393,394] For the tests, a 0.6 m dia. and 1.5 m long Humboldt Wedag machine was chosen. This was tested at the Morrison Busty coal preparation plant, North East Area. At the original speed of 1,537 rev/min the centrifugal force was 792 times the gravitational acceleration and the scroll ran 17 rev/min faster than the bowl. Improvements in dewatering were made by increasing the bowl speed to 1,661 rev/min and by increasing the pool depth from 50 mm to 62.5 mm and by recirculating the screen effluent. With these modifications, 95% of the solids were recovered at 23% moisture content and at a feed rate of 9 t/h. The ash content of the centrifuge product was 1% lower than that of the feed. When compared with vacuum filters operating on similar feeds the moisture content of the filter cake was 27% as compared with 23% of the screen bowl centrifuge product. The solids recovery of the filter is expected to be higher. It is an advantage that the ash content of the centrifuge product is 1% lower than that of the filter cake. The centrifuge occupies only one quarter of the floor area occupied by the filter and only one tenth of the volume required for the filter and its ancillary equipment. Power and capital costs are about the same. The screen bowl

centrifuge tested has proved to be a satisfactory machine for dewatering froth floated fines. It requires regular annual maintenance, but very little other attention.

Screen bowl centrifuges have been used with excellent results in the coal preparation plants of Ruhrkohle AG.[395] In 1979, a screen bowl centrifuge with 0.9 m diameter and a 1.8 m long rotor was installed for dewatering fine coal at 10 t/h. After seven months continuous use without any failure, wear was found to be minimal. Three larger, 1.1 m by 3.3 m, machines were installed, each with a capacity of 18t/h. The larger model was later lined with ceramic plates, including the screw conveyor, cylindrical conical sections and screens. The service life of this model is expected to be between 8000 and 10000 hours. The solids recovery is about 95%. Cost analyses indicate that screen bowl centrifuges show considerable economic advantages over vacuum and pressure filtration.

Green discusses the use of solid and screen bowl centrifuges and lists models, manufacturers and technical information.[396] Since the introduction of screen bowl centrifuges their performances have been continuously improved.[397] The newer machines are able to dewater wider size ranges and give longer service life. Test comparisons with alternative methods show that screen bowl centrifuges are cost effective and technically efficient. Another paper discusses recent developments in the dewatering of small and fine coals in the F.R.G..[398] The operations of centrifuge types including solid-bowl, screen-bowl, pusher, shaker, basket and decanter are described. Drum, double drum, disc and belt type vacuum filters, conventional and band filter presses are also dealt with. The fundamental principles of centrifugal sedimentation and filtration are discussed by Svarovsky,[399] Zeitsch[400] and Purchas.[401]

A recent survey of centrifuges of various types includes brief descriptions, their main areas of application and their expected capacities and performance.[402] The centrifuges are listed according to manufacturer with brief notes on the models available.

Experimental studies on a dewatering device called Dehydrum have been carried out on the waste water from coal preparation plants.[403] The operation of the apparatus is based on pellet flocculation. With the use of agitation, random flocs are converted into much more compact pellet flocs. The conditions necessary for the formation of pellet flocs are strong polymer bridges between particles and the presence of locally uneven fluctuating forces to cause rearrangement of particles in the floc. The apparatus tested was 2.4m diameter by 6.7 m long, rotating at a peripheral speed of 0.42 m/min. The waste water contained 3% solids by weight, 47% of which was below 5 μm in size. A partially hydrolysed polyacrylamide flocculant was used and some calcium chloride and alum were also added as coagulants. At a feed rate of 3.5 tonne/h of thickener underflow, dewatered cake of 50% solids concentration was produced and the final effluent had a solids content below 150 ppm. The chemical costs evaluated for a waste water discharge rate of 400 m^3/h were between $42-$52/h in 1977. The separator is said to have high treatment capacity, easy operation and maintenance, low initial cost, moderate running costs and low solid content of the final effluent.

A new device, called the Dehydrator, was developed in Poland for dewatering polymer flocculated fines.[404] The Dehydrator uses relatively high flocculant dosages of 0.2 to 0.3 kg/t of feed. It forms floc pellets in a rotating cylinder of 1 m dia. and 6 m long, inclined at 30°. The rotational speed is 3 to 9 rev/min. The cylinder incorporates a spiral for the transport of flocs. The lower half of the cylinder surface is solid while the upper half is made from screen mesh to allow drainage. The throughput of the Dehydrator is 30t/h and its product is said to contain 3% to 8% less moisture than it would in vacuum filtration. Capital and operating costs of the Dehydrator are low, the cost of the flocculant being the main item of expenditure.

2.19 Filtration

Filtration of fine mineral and coal slurries is carried out either on continuous drum, disc or belt type vacuum filters or in batch type pressure filtration. Both vacuum and pressure filtration may apply compression to the cake for improved moisture reduction.

Continuous vacuum filters have several advantages, compared with pressure filtration equipment.[405] The feed slurry to a vacuum filter can be delivered by gravity flow or by low pressure pumps. Vacuum filtration equipment does not have to withstand high pressures and thus may be constructed inexpensively. As filter cake removal is easy, short filtration times may be chosen, taking advantage of the low resistance of thin filter cakes and the associated high filtration rates. There are also some disadvantages. Continuous vacuum filtration requires a vacuum system consisting of vacuum pumps, filtrate pumps and supplementary accessories. Vacuum filters are unsuitable for high cake resistance and for highly compressible cakes. These filters are sensitive to changes in feed rate, consistency and feed solids characteristics. Despite these disadvantages, continuous vacuum filters handle a greater tonnage of solids than do all other types of filters combined. Vacuum filters include the common rotary drum filters with various cake discharge arrangements (string, knife, roller belt, precoat), internal drum, top feed drum, rotary disc, horizontal table and horizontal belt filters. The operation of the various types of filters is discussed and their selection for different slurry characteristics is outlined.

A review of current filtration equipment describes modern filtration practices.[406] The belt discharge type rotary drum filter is usually equipped with belt washing and is an efficient and frequently employed filter. Horizontal belt vacuum filters can handle relatively coarse feed. They are designed in three forms, with transporter belt, with stationary vacuum pans and with moving vacuum pans. Pressure filters are also included in the article and a list of manufacturers of filtration equipment is also given.

The double drum rotary vacuum filter has been used successfully over several years for the dewatering of raw slurry in a coal preparation plant in the Ruhr, West Germany.[407] The double drum filter consists of two filter drums spaced closely with their axes parallel. Side plates, with sliding seals fitted at the drum ends, seal against the drum edges to form a feed vessel between the drums into which the pulp is fed from the top. The filter drums, rotating in opposite directions, filter the pulp in the cake build-up zone between the drums and carry the cake up into the dewatering zones. The cake is then removed by scrapers from each drum. This arrangement dispenses with the feed tank, agitator and agitator drive required by conventional rotary drum filters. The double drum unit manufactured by KHD Humboldt Wedag AG, consists of two 2.1 m dia. by 5 m long drums. Double drum filters operate best with relatively coarse slurries, in which differential sedimentation of the coarse particles occurs and forms a high permeability base layer of the filter cake, as happens in top bed filters. Consequently, double drum filters have high specifid filtration rates of around 1 t/m^2h and give lower filter cake moisture content. The maintenance on drums and filter cloths is said to be easy due to the open and accessible design.

At the Bentinck colliery coal preparation plant, Nottinghamshire, the vacuum filter installation system has recently been redesigned to eliminate filtrate pumps.[408] The newly installed pneumatic filtrate discharge equipment consists of two tanks fitted with pneumatic valves, a control box with electropneumatic solenoids and timing devices. When in use filtrate is delivered to tank A from where it passes to tank B. When the filtrate in tank A reaches a predetermined level tank B is isolated from tank A. Tank B is pressurised by compressed air between 200 and 350 kPa (30 to 50 lb/in^2), and its filtrate is discharged. On reconnection tank A drains into tank B and the cycle is repeated. This vacuum discharge system is easy to install. By eliminating barometric legs and filtrate pumps, the new system requires only one floor height and offers savings in capital costs.

Oil assisted filtration of fine coal involves the introduction of up to 3% of oil into the fine coal slurry before filtration.[409,410] The oil droplets coagulate the coal and improve filtration performance. Following the addition of 1.5% of oil, the moisture content of filter cake dropped from 26% to 16% and the filtration rate increased tenfold. The ash content of the filter cake was also reduced due to the selective adsorption of oil on coal, leaving the fine clay dispersed. The dispersed clay was able to pass through the filter cake and cloth with the filtrate.

Steam assisted vacuum filtration is a valuable technique for the reduction of moisture content of coal filter cakes.[411] A steam hood is mounted above the rotary drum to supply steam which increases cake temperature and reduces the surface tension of the filtrate. Use of this technique reduces moisture content of cake by 5% - 9%. Approximately 1.2t of steam is required to remove 1t of water. Another advantage is the improved handleability of the clean coal.

Dorr-Oliver has introduced the Electrofilter, a vacuum filter which also uses electrosmosis.[406] This filter contain electrodes, the cathode covered by the filter cloth and the anode situated in the slurry. The vacuum maintained over the filter cloth pulls the filtrate through while the solids migrate towards the anodes and deposit a cake which can be removed mechanically.

Squeezing action in filtration and dewatering of coagulated and particularly flocculated, slurries has been known to be important for a long time, but only in the past few years have industrial filters been designed to take full advantage of squeezing. In normal filtration a considerable pressure drop may occur across the filter cake. This pressure drop builds up from the top of the cake and, therefore, at the top there is no squeezing action. Thus, with compressible cakes, the top of the cake is less compact and contains more moisture. When squeezing action is applied to a previously formed filter cake, the cake may be compressed and further moisture may be removed. This action may be utilised in continuously operating belt filter presses and in intermittently operating plate type filter presses.

A production model of a belt filter press from Parkson Corp., Florida, was mounted in a trailer and used in various coal preparation plants for coal refuse dewatering.[142, 413] A schematic diagram of the belt filter press is shown in

Figure 60. The flocculated feed is first drained on the belt and then pressed in two stages, at low pressure followed by high. Tests carried out show that when operating on a feed of 20% solids by weight, the feed belt capacity is around 9 m^3 pulp/h or 1.8t solids/h, while the moisture content of the cake is 25% to 30% by weight. The flocculant dosages required were quite high, between 0.2 and 0.34 kg/t costing US$0.79 to $1.34/t of solids for reagents. However, power consumption was only 3.7 kW for the belt filter against about 75 kW for vacuum disc filters.

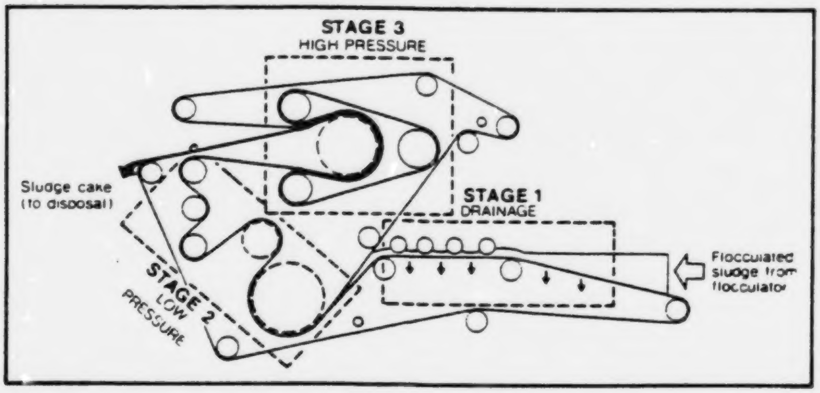


FIGURE 60: Schematic diagram of the belt filter press.

Cake squeezing action has been used in conventional filter presses equipped with membrane filter plates, such as those offered by Johnson-Progress. Some of the latest pressure filters are designed in a vertical configuration. Larox of Finland makes vertical automatic pressure filters which incorporate cake squeezing by rubber diaphragms.[414] The filter plates are arranged horizontally and the continuous belt of filter cloth zig zags between the plates. Each plate has a pressure diaphragm which squeezes the cake and the chambers in which the filter cake forms are approximately

45 mm deep. The filtration cycle begins when the filter is closed and the plates are tightly pressed against each other and has the following stages:

1. Filtration. All chambers are fed simultaneously with slurry at high pressure, the filtrate passes through the filter cloth and the polypropylene backing grid and then drains through hoses connected to filter plates.

2. Pressing. High pressure water presses the rubber diaphragms onto the cake to reduce the residual moisture. Following pressing, cake washing may be carried out if required, after which a second pressing is needed.

3. Air blow. Dewatering is completed by blowing compressed air through the cake to make it as dry as mechanically possible.

4. Cake discharge. The filter opens with the same clearance between all plates and the filter cloth, running over rollers, pulls the cake out to either side of the machine. As the cloth passes the rollers most of the cake falls off and any remaining is removed by scrapers.

The cycle is repeated automatically. Blocking of the filter cloth by particles is minimised as the direction of flow of the filtrate through the cloth changes after each cycle, thus washing the cloth clean. Additional washing is applied to both sides of the cloth by water sprays as the cloth moves between filtration cycles. Larox pressure filters are available with six to twenty plates in two plate sizes, 500 x 800 mm and 900 x 1,750 mm.

Larox horizontal pressure filters have been installed in a Finnish nickel concentrator[415] and a phosphate plant.[416] Other concentrators using these filters include the copper concentrator of Noranda Mines,[417] Brunswick Mining [417,418] and Inspiration Copper Company, Arizona.[419] All these installations show reduced moisture contents and inproved economies. Moisture contents of copper concentrates produced by the Larox horizontal pressure filters are usually between 8% - 12%. At Inspiration Copper, bending of filter plates was experienced due to blockage of slurry feed hoses by coarse particles, glass etc. This emphasises the importance of removing coarse materials from filter feeds.

Larox horizontal pressure filters may be useful for dewatering coal fines to reduce moisture contents. The lower density of coal compared with high density mineral concentrates will make the moisture content of coal filter cakes considerably higher.

Larox Oy has also introduced a vertical automatic chamber filter, model CF, which has optional diaphragm pressing.[420] It comes with a plate size of 1.320 mm x 1.970 mm, with up to 80 plates and 208 m^2 filtration area. The largest unit has a 6 m^3 chamber volume, weighs 65 t, and has a floor area requirement of 80 m^2.

A Lasta automatic filter press, manufactured by IMPCO Division of the U.S.-based Ingersoll Rand Corp., has been installed at the Butte concentrator of Anaconda Minerals Co., Montana. U.S.A.[421,422] The filter dewaters copper concentrate from 65% solids to 8% moisture content in 15 to 20 min cycles. During each cycle it yields 4.5 t to 5 t of cake. The capital cost of the pressure filter was about half that of the drum or disc filter and thermal drying equipment capable of the same throughput. Initial estimates show that the operating cost of the pressure filter is only 40% of those of alternative dewatering methods. In mid-December 1981, another Lasta automatic filter press was installed at Patino Mines' Northgate concentrator, Chibougamau, Quebec.[423] Previously, disc filters were used which gave a cake with 12.5% moisture content. With the Lasta filter, operating at 10t to 12t/h throughput, the moisture content was reduced to about 8% which has produced a monthly saving of $C26,000

A new filter cake dewatering aid, the Drimax 1938 has recently been announced by Allied Colloids, as a new addition to its Drimax range.[424] This dewatering aid is said to modify the surface characteristics of solids to bring about considerable reductions in filter cake moisture content. With some slurries as much as 28% moisture reduction has been observed. Main areas of application are vacuum and pressure filtration, centrifuging and gravity drainage. It also aids the removal of cake from the filter cloth and remains effective in either acidic or alkaline conditions. Reduction of moisture content of filtered mineral concentrates results in either lower drying or lower transport cost, whereas improved filtrate recovery is advantageous in hydrometallurgical applications.

2.20 Tailings Disposal

Tailings disposal has received increasing attention in the last two decades. This may be attributed to a number of factors of which the more important are: increased concern for safety following a number of disasters, greater sensitivity towards the environmental impact of mining and significantly increased volumes of tailings resulting from the combination of exponential growth and steadily decreasing ore grades.

Downing and Stocks discuss tailings disposal methods and the associated environmental problems[425,426] The disposal of coal preparation tailing, particularly of fines, is often an expensive and complex process. Impoundment is the most widely used method, although the stability of tips is frequently unpredictable. Underground disposal is labour intensive.[427] Wide scale application of mechanical dewatering is limited by economic factors. Vacuum disc filtration is not usually satisfactory because of low filtration rates, blocking of filter cloth, high moisture content of cake and high maintenance costs. Pressure filtration is accepted in Europe but not so far in North America. Flocculation of feed is usually necessary for successful operation, although in some plants coagulated feeds give good results. The use of belt presses is a more recent practice. Centrifugation in solid bowl centrifuges is only of limited success. Chemical stabilisation with certain additives has been reported to produce a residual solids mass of substantial strength. Electrokinetic dewatering which utilises electrophoresis and electroosmosis has been developed by the U.S. Bureau of Mines. Electrophoresis is the movement of electrically charged particles in a stationary liquid by an external electrical potential. Electroosmosis is the migration of liquid through a stationary porous bed of particles, caused by an external electrical potential. In electrokinetic dewatering, electrophoresis results in a consolidation of suspended solids. Once consolidation is complete, further dewatering is achieved by electroosmosis. Electrokinetic dewatering has been used in the United States, but due to its high electrical energy consumption, the process is costly.

Fine coal washery tailings may be stabilised by the addition of cement.[428,429] The underflow of Deep Cone Thickeners may be as high as 60% solids by weight and is, therefore, ideal for stabilisation. In the stabilisation process the thickener underflow is collected in a 10t capacity container From the bottom of this container the tailings are withdrawn by two screw conveyors and deposited into a continuous cement mixer. A variable speed belt conveyor introduces cement at a predetermined rate into the cement mixer. Usually 40 kg cement/t of tailing is sufficient. Coarse tailing is added to the mixture before it is deposited onto the tip. About 50t of cement is used weekly to stabilise 1,250t of thickened tailing containing 750t of solids.

2.21 Reclamation

More positive uses for mill tailings have recently been seriously considered. The recovery of valuable minerals from old tailings is now feasible due to improved technology and to economic changes allowing profitable recovery of some of the residual values.[430] Minerals rejected as waste at the time of mining may acquire a value which warrants reworking the tailings, due to changes in mineral requirements as with fluorspar and uranium. Another use for tailings is as backfill. The civil engineering industry in its major projects, requires large volumes of backfill materials. Not all tailings are suitable for this purpose; some may be too fine, thixotropic or may have undesirable chemical properties.

Reclamation of coal from mining waste by Scrubber Mills and autogenous media cyclones as used in North Derbyshire, U.K. is outlined.[431] The process gives clean coal and a solid waste which is easily disposable. Green discusses the reclamation of anthracite in Luzerne County, Pennsylvania.[432] This operation treats the waste feed at 500t/h in a dense medium drum and dense medium cyclones.

The viability of beneficiating Canadian waste coals has been studied.[433] It was found that to obtain clean coal of sufficiently low ash content, fine grinding (finer than 25 μm) had to be employed which is not an economic proposition at present.

2.22 Coal Washability Characteristics

Float and sink analysis is commonly used for the prediction of coal washability results. Washability determinations are included in the Australian Standard (AS 1661-1979) which recommends the use of five curves. 1. the cumulative floats curve, 2. the cumulative sinks curve, 3. the relative density curve, 4. the instantaneous ash curve and 5. the ±0.1 relative density curve, Figure 61. The characteristics of washability curves which categorise a coal are: 1. The ash in raw coal - intercept of curve 1 with the bottom axis. 2. Minimum possible ash in floats - extrapolated intercept of curve 1 with top axis. 3. Maximum possible ash in sinks - entrapolated intercept of curve two with bottom axis. 4. The shape and position of curves on the graph. Plateaux on curves 4 and 5 show the range of separation density in which there is a minimum quantity of near gravity material, corresponding to a region of easy separation.[434] The distance of the plateau on curve 4 from the bottom of the graph indicates the proportion of reject material contained in the raw coal. The ease of separation may be evaluated by consideration of the relative density, the instantaneous ash and the ±0.1 relative density curves (Curves 3, 4 and 5). If the curves are sharply angled, as in Figure 61, separation will be easy at relative densities above the value at which the curves approach the horizontal. Where the lines are gently curved throughout their lengths as in Figure 62, separation is likely to be very difficult. Four main categories of washability characteristics are recognized. In the construction and interpretation of washability curves certain points should be observed. It is essential that float and sink tests are carried out over a sufficient

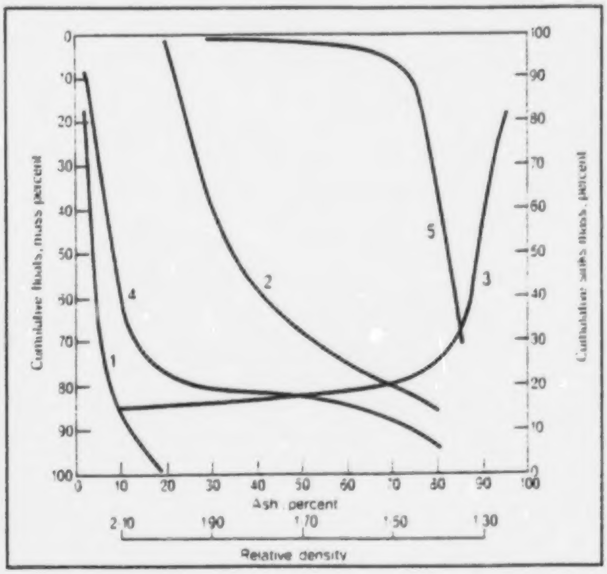


FIGURE 61: Washability characteristics of a coal which is very easy to separate. Category 1

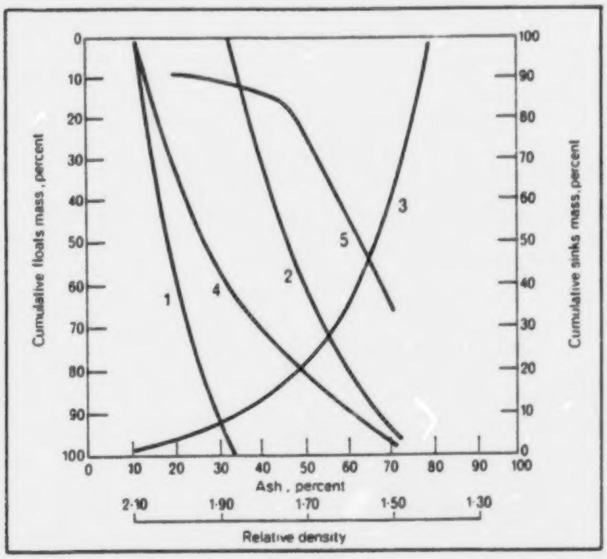


FIGURE 62: Washability characteristics of a coal which is excessively difficult to separate. Category IV.

number and range of densities and on representative samples. Washability results are useful in the interpretation of separation performance which relies entirely on density differences but have little meaning if applied to other forms of separation such as flotation. Although washability curves predict ideal separation results, industrial operations will produce lower yield at a given ash assay or a higher clean coal ash at a theoretical yield. Washability curves give no indication of the coking properties of a coal. However, the ash content of coking coals is usually between 8% and 10% which determines the required separation density. Crushing is often used prior to separation to reduce the ash content of clean coal. On the whole Australian coals do not exhibit as much reduction in ash content at smaller sizes as do British and North American coals. For steaming coals, adequate upgrading can be achieved with jigs on the coarser sizes and water only cyclones for the fines. For coking coals the separation frequently has to be made on a difficult part of the washability curve, requiring precision which can only be achieved with dense medium separation. In this case froth flotation is used for the fines. The proportion treated by froth flotation is increasing due to the trend towards reduced maximum size. This may simplify plant flowsheets by allowing the use of dense medium cyclones alone, dispensing with dense medium baths. Economically this may be undesirable as the minus 0.5 mm fines have to be froth floated at treatment costs of about two and a half times those of dense medium flotation.

Application of computer methods to coal washability calculations reduces the tedious work and broadens the capabilities of coal preparation engineers in predicting coal processing results.[435-437] Since these methods reduce the amount of numerical work to the minimum, the cleaning performance may be studied in detail and optimised.

The performance of dense medium cleaning devices is customarily characterised by the distribution curve, in which the percentage of feed reporting to clean coal is plotted against specific gravity. For a certain separator and a given feed a family of distribution curves can be obtained, each having its own specific gravity of separation, as shown in Figure 63 (left). Gottfried proposes a technique for combining these distribution curves at different specific gravities of separation into a generalised distribution curve which is independent of the specific gravity of separation, as given in Figure 63. (right).[438] In the generalised distribution curve the reduced specific gravity, x, is defined as $x = \rho/\rho_s$, where ρ is specific gravity as a variable and ρ_s is the specific gravity of separation. From the generalised distribution curve the generalised probable error, GPE, may be calculated. Since the GPE is independent of the specific gravity of separation, it appears that for a given separator and feed, with the feed falling within a given size fraction, the probable error, PE, is proportional to the specific gravity of separation, i.e. $PE = \rho_s$ GPE.

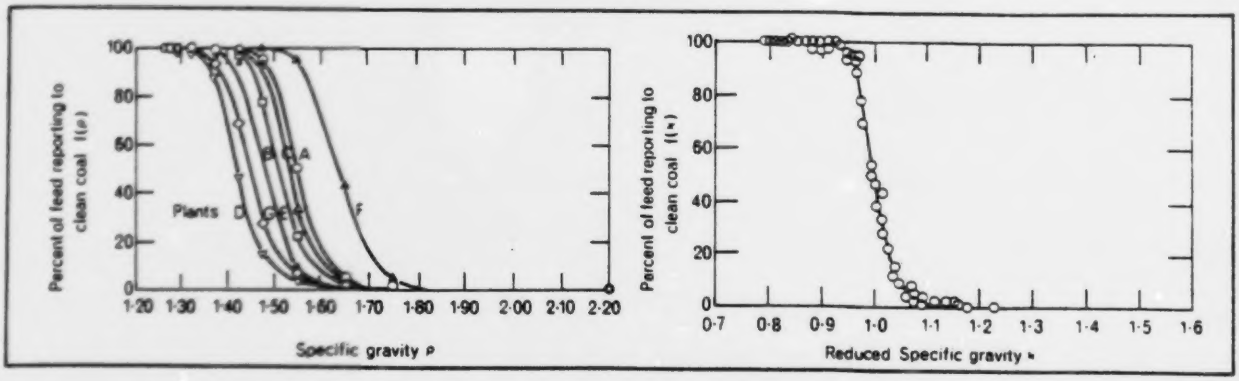


FIGURE 63: Conventional distribution curves for a dense medium cyclone (left), the generalized distribution curve (right).

2.23 Process Modelling

During the last fifteen years, process modelling and simulation have become important in mineral processing as they are essential for effective plant control. At the 14th International Mineral Processing Congress a session was devoted to the topic, during which twelve papers were discussed. Process simulation can also be extremely valuable in evaluating the effects of operating variables, optimisation of plant performance and for the optimal design of plant flowsheets. The diagnostic use of simulation is a more recent application of great potential. Every process simulation project must begin with data acquisition, in which data is collected systematically and meaningfully.[439] Careful sampling is most important. Repetitive sampling at regular intervals provides information on the short and long-term variations occurring in the circuit. One example is, where such sampling of a hydrocyclone overflow diagnosed instability in the cyclone feeding system. When the results were plotted as frequency against solids concentration of cyclone overflow a bimodal frequency distribution was obtained pointing to cyclic fluctuation. Analysis of raw data can indicate a faulty control strategy. Variation of ore hardness in grind will create over and under-grinding unless the feed rate or the circulating load of the circuit is controlled. The raw data have random errors and also reflect the variations and fluctuations occurring in the circuit. So, at any time, the data collected will not satisfy materials balances. However, by collecting excess data, it is possible to reduce random errors and to produce consistent data. Modern computer techniques for materials balance of mineral processing circuits are now available.[440] Some include sensitivity analysis to evaluate the reliability of results.

Modelling is the representation, in a mathematical form, of the process behaviour as a function of operating variables.

There are two main types of models. (1) Empirical models simply represent the process behaviour in empirical relationships. (2) Phenomenological models are based, at least partially, on physical laws relating to the process. The second type is far more informative and aids correct interpretation of operating data. The empirical regression type models can sometimes mask true phenomena due to the interdependence of operating variables. Mathematical modelling and its application is a complex subject, often beyond the understanding of most mineral processing engineers. But those who specialise in this area are in possession of a powerful tool which, when properly applied, can significantly improve the operating performance of a plant. The correct choice of modelling variables is important. In a grinding circuit the operating set points may be residence time in mill, classifier cut size and circulating load. Performance parameters of the mill may be mill hold-up, pulp viscosity, ore grindability; and of the classifier, may be partition sharpness and by-passing coefficient. These concepts are useful for the interpretation of industrial data corresponding to various operating conditions, because they allow the determination of the exact cause of the observed effect. For phenomenological modelling, the various unit operations must be presented in mathematical relationships as governed by certain physical laws. There is a growing collection of these models for the various operations, and their accuracy in simulating the process is expected to improve. It is important that the model should adequately describe the process without being too complicated for use.[441] The more important models in use are for crushing, grinding, hydrocyclone classification and flotation. Some models relating to flotation were discussed earlier in the section on flotation. Another important requirement is that the model constants should be easily determinable from test data.

The computer program links the models of the various machines in a circuit and is used to study the effect of the operating variables. Simulation can be very informative because it is possible to study the effect of a single variable at a time, and because simulation does not have any fluctuations or noise. For these reasons, simulation results are more easily interpreted than experimental results and can lead more directly to the correct understanding of cause and effect. For any circuit, there is a certain number of operating variables which must be given to define the steady state of the circuit. When such a steady state exists, it is unique. By changing the operating variables, the circuit is simulated. When the number of constraints is less than the independent variables, then there is no unique possible steady state and the function can be minimised or maximised to find the best steady state value of the circuits. This manipulation is termed optimal tuning. Dynamic simulation can predict the result of transient changes in the circuit.

Dynamic simulation is also useful for the evaluation of the sensitivity of the circuit to sudden changes in operating variables. Simulation can be important in the optimal design of a circuit, as it aids the determination of the optimum number and size of machines to attain a defined objective. It is useful when existing circuits are to be modified. The practical benefits accruing from process analysis, modelling and simulation are substantial and should be fully utilized in coal preparation:

Erasmus developed a mathematical model for the prediction of dense medium separation performance.[442]. He used his mathematical model to compare the performances of 140 dense medium cyclones and 72 Drewboy dense medium baths operated over the separation density range of 1.36g/cm^3 - 1.68 g/cm^3. Erasmus found that considerable variations existed in separation performance. The error area increased with increasing separation density. As the separation density increased, the partition curves of dense medium baths became more symmetrical while for dense medium cyclones they became more asymmetrical.

Computer programs have been developed for the simulation of mineral processing circuits. One such program is called BILMAT and was developed by Laval University, Quebec.[443] Another program called FLEXMET was written for evaluating circuits in mineral processing and extractive metallurgy, by Fluor Mining and Metals Inc. in collaboration with the University of Arizona.[444] The SPOC project (Simulated Processing of Ore and Coal) was initiated in April 1980 by CANMET.[445] Several software packages are now available under sampling, material balance, model calibration and process simulation. These material balance and simulation programs should aid the optimization of coal preparation plants.

2.24 Process Control

Automatic on-line control has been used in mineral processing plants since 1968. In Canada, Lake Dufault copper-zinc concentrator was one of the first to use on-line digital control. The application of computerized process control in new mineral processing plants has become common practice and many older plants have also been equipped. Although ash determination has been used in coal preparation since the early 1970s it was mainly used for the blending of clean and raw coals to optimize the ash content of power station fuel.[446] There are differences between the applicability of process control to coal preparation and to mineral processing. Commonly, in mineral processing the ore has to be relatively finely ground and the separation takes place at finer sizes and thus assaying for the valuable component is easier. Coarse feeds require large samples to be representative and, therefore, mineral feeds are analysed after grinding. In coarse dense medium separation of coal, it would be difficult to control the separation directly from the ash content of the products. Ash content monitoring of coarse coal needs continuous size reduction of a portion of the product for continuous ash analysis. This introduces an undersirable time delay. Fine coal cleaning is more amenable to on-stream control and with improved ash analyzers it should not be difficult to achieve flotation circuit control similar to that used in mineral processing. In these circuits the flotation reagent addition and often the aeration rate and pulp level in the cell are controlled from on-stream measurements through the use of some mathematical model.

Mineral grinding circuits are often the greatest energy consumers in the plant and, therefore, their efficient control is important. The size distribution of the ground product also governs mineral recoveries and fines losses. In normal coal preparation practice the coal is not ground. But in intensive cleaning and for coal-liquid mixtures grinding will become more significant.

The instrumentation and process control of mineral processing plants is well documented.[447-449] Ulsoy and Sastry give an extensive list of references and describe basic control strategies.[448] The computer control of mineral and coal flotation circuits is discussed.[308,450] Control is needed in plants because of disturbances of two kinds:[451]

1. Variations of feed characteristics.
2. Variations in plant operation.

The first type of disturbance may be considerably reduced by blending the feed to the plant, resulting in improved plant operation. Blending at finer particle sizes is much more effective, since at coarse sizes large quantities of feed must be blended to even out variations. In mineral processing the final blending is carried out at the fine ore stage, below 20 mm particle size.

The chemical composition, or more usually the metal content, of an ore stream is most often determined by X-ray fluorescence analysis. X-ray fluorescence analysis utilizes the characteristic X-rays emitted by elements when they are irradiated by X-rays or gamma rays. An X-ray fluorescence analyzer system, such as the Outokumpu Courier 300, is usually capable of measuring five to six elements in as many slurry streams as required. A central analyzer assays the slurry streams in sequence, taking about twenty seconds for each stream, during which the elements are analyzed simultaneously. An alternative method uses in-stream X-ray fluorescence probes submerged in the slurry streams, e.g. the Outokumpu Minexan 202. With in-stream probes no sampling is needed but each stream must have a separate analyzer. The Minexan 202 uses a radioisotope source and a solid state detector and is able to analyze up to five elements with atomic numbers above calcium.[452] The probe also measures the pulp density. Three probes are regarded as a practical minimum for successful monitoring and control of a process. In larger plants it may be more economical to employ one central analyzer.

The National Coal Board-Sortex ash monitor has been well tested and is installed in most new coal preparation plants in the U.K. This monitor measures the ash content of a continuous dry sample crushed in a swing hammer mill.[446] The instrument uses the back scattered rays from a plutonium

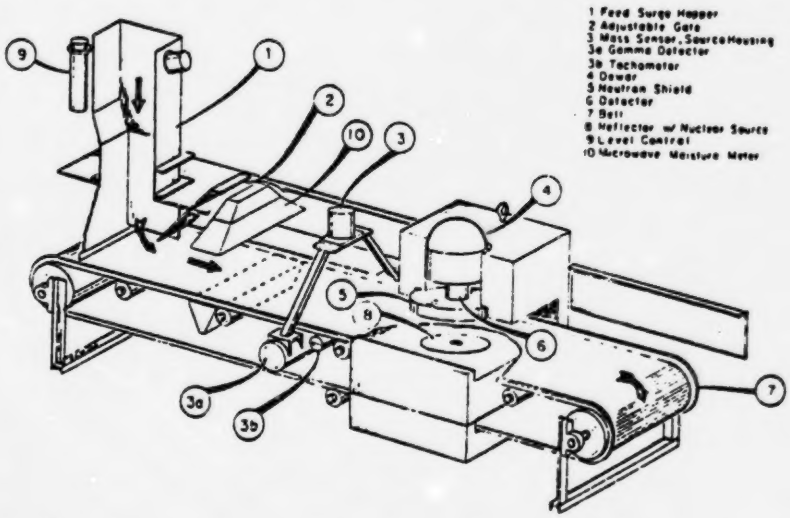
238 isotope. Coal, containing carbon and hydrogen scatters much less radiation than shale containing silicon, aluminum and iron. Although this monitor is used for blending power station fuel to the required ash content, it could also be used for the control of coal cleaning circuits.

The CONAC analyzer developed jointly by Science Application Inc. and the McNally Group through its subsidiary, Kennedy Van Saun Corporation, employs prompt neutron activation analysis, (Figure 64). It is capable of measuring carbon, sulphur, hydrogen and the principal elements of ash.[453] The moisture content is also measured by an electromagnet technique. The advantages of the system are said to be:

1. Provides non-destructive coal assays.
2. Can be applied to coal streams up to 30 t/h which reduces sampling errors.
3. Has fast response.
4. The analysis is almost independent of coal type and particle size.

Measurement of ash content of coal slurries has also been developed.[454,455] These techniques use the measurement of scattered radiation and a gamma guage for the measurement of solids concentration.

Other essential measurements are flow, level and pulp solids concentration. Flow may be measured by ultrasonic and Doppler shift flowmeters.[456] Level measurements may be contact type or the non-contact type such as nuclear or ultrasonic. For the measurement of solids concentration a gamma ray source and a detector attached on either side


1 Feed Surge Hopper
2 Adjustable Gate
3 Mass Sensor, Source Housing
3a Gamma Detector
3b Tachometer
4 Dewar
5 Neutron Shield
6 Detector
7 Belt
8 Reflector w/ Nuclear Source
9 Level Control
10 Microwave Moisture Meter
9
1
2
10
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8
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3a
3b
7

FIGURE 64: Schematic view of the CONAC ash analyzer.

of the slurry transport pipe are commonly used.

Particle size analysis for coarse material travelling on a conveyor belt has been developed. This optical system is suitable for particle sizes between 300 mm and 3 mm.[457]

The Armco Autometrics PSM 400 Particle Size Monitor provides continuous measurement of particle size and solids concentration of slurries.[458] The accuracy of size determination is typically ±1.5% of the actual cumulative percent passing.

Vibration and noise analysis is not directly used in process control but it is a very useful technique to prevent costly breakdowns. Careful analysis of frequency components present in the signal permits positive identification of iminent problems.[459] For example, vibration caused by a damaged gear tooth will have a definite frequency which is directly related to the rotational speed and number of gear teeth. The vital part of the vibration measuring equipment is the transducer which converts mechanical energy to electrical signals. Small changes in the mechanical condition of a machine produce significant changes in vibration levels. There are two aspects of vibration amplitude to observe: its absolute level and its trend. The absolute level of amplitude is typical for a certain machine. Unusually high levels indicate a severe defect in the machine. The overall trend of the vibration level is equally important. If it remains essentially constant over a long period (months) the machine is in good condition. A rising trend indicates developing defects. Vibration analysis is a powerful tool in preventive maintenance and its use should widen. It should include on-line monitoring and computer analysis to detect developing faults in major machinery well before breakdown occurs.

Computer based process control is being increasingly installed in coal preparation plants. In the U.K. at least fifteen coal preparation plants used computer control in 1982.[460,461] In the U.S. the newer plants are also controlled by computers, microprocessors and programmable logic controllers.[462,463] and in Australia the trend is similar.[464]

The accumulated experience in process control of mineral plants should be tapped to help evaluate coal preparation process control strategies. Although some of the process control developed for minerals is not applicable to coal (e.g. comminution circuit control) there is still a good proportion to be considered, but with a certain caution as coal is not just another mineral.

2.25 Plant Design

Major trends in mineral processing plant design include the use of large machines, simplification of process flow diagrams, construction of crushing and grinding sections outside the plant building and the application of computerized on-line process control.[465] There are similar trends in coal preparation. The general principles of plant design are covered in an AIME volume, entitled Mineral Processing Plant Design.[50]

The use of well selected large units of equipment in high capacity plants is discussed by Hillman.[466] Capacities of modern plants are currently between 800-1300t/h and these plants benefit from the installation of large equipment. For example, a single 3.7 m wide dense medium bath would be sufficient to treat the 150 mm - 12 mm fraction of 1500t/h run-of-mine coal. For large equipment, the capital costs per tonne treated is lower than for smaller equipment and operating costs are also reduced as larger units are often more efficient.[467] Fewer large units simplify plant layout, but in coal preparation plants built with single lines of large capacity units break-downs are usually costly. Therefore, these plants must be built with reliable equipment and process control and adequately maintained.

Modular coal preparation plants, which may be rapidly assembled, are offered by some manufacturers. Envirotech Coal Services of Beckley, West Virginia, build modular design plants in six months or less.[468-470] The 200t/h units, which may be expanded, include dense medium separation and froth flotation (when required) with a closed water circuit. The modules are assembled on site and a prefabricated steel building is erected around them. Since the building is free standing, vibration and noise transmission are reduced. The raw coal feed is cleaned to zero size in dense medium cyclones. The minus 0.5 mm clean coal passing through the rinsing screen with the dense medium is recovered in magnetic separators as a non-magnetic product. Both the fine and the coarse clean coal are dewatered in centrifuges. The fine tailing from the refuse rinsing screen is separated in another magnetic separator as the non-magnetic product, thickened and vacuum filtered. The plant is equipped with the necessary automation and control and full process automation is optional.

McNally Pittsburg Manufacturing Inc. installed a modular coal preparation plant for Lanny P'Pool of Nemo Coal Inc., Moberley, in 11 weeks.[471] This is probably the fastest construction of a coal preparation plant. Each of the modules is a rectangular steel framed box measuring 9.1 x 3.3 x 3.7 m high. The plant consists of five units stacked on top of each other with two units stacked beside. The plant treats 38 to 6 mm raw feed at 200t/h producing clean coal with 5% to 7% ash and 2.5% to 4% sulphur content.

The 550t/h coal preparation plant of the Windsor Power House Coal Co., near Wheeling, West Virginia, started operation in 1981.[462] The plant, designed by Norton and Hambleton, incorporates many interesting features. A departure from traditional design is that the major pieces of equipment are spread horizontally and thus the building has a low profile. Conventional coal preparation plants are high to utilize gravity flow and this configuration makes them more costly. At Windsor the equipment is installed on steel platforms rather than on the floors of the building. This arrangement allows the erection of a cheap, commercially prefabricated building over the plant, because the building only carries its own weight. Every piece of equipment, except the centrifuges and the filter press, may be moved to a 7.6 m x 6.1 m service bay for maintenance work. In spite of the low height of the design, the floor area of the plant building is only 30.5 m x 24.4 m. The interior is not cramped and all equipment can be seen from the control room. The construction costs were lower because much of the equipment and building was factory made. The design and construction of the plant cost $US 14 million, which is about 2/3 of the normal cost of a plant of this size and complexity.

The circuit consists of dense medium separation in a Wemco drum to produce clean coal, middlings and refuse. The middling product is crushed and returned to the raw coal screen for further treatment. The coarse clean coal is crushed and graded while the coarse refuse is discarded. The fines from the raw coal screen are deslimed and separated in dense medium cyclones to obtain clean coal and refuse. An important feature of this plant is that the desliming screen underflow is pumped to a thickener which removed the high ash slimes into the overflow to be discarded in the refuse thickener. The slimes removal before flotation is beneficial. The deslimed thickener underflow goes to a conditioner and to flotation cells to recover the clean coal fines. The frothed coal is dewatered in screen bowl centrifuges and joins the coarser clean coal products. The fine refuse is thickened and pressure filtered. The Windsor plant is fully automated. Start-up and shutdown are carried out by a programmable controller which also looks after circuit failures and deals with problems such as too high centrifuge torque, indicating overload, or insufficient magnetite in the dense medium circuit.

Another new coal preparation plant, based on a similar philosophy, was designed and built by KHD Humboldt Wedag AG, at the Neu-Monopol mine at Bergkamen in the Ruhr, West Germany.[472,473] The plant handles 300t/h coking coal and 700t/h steam raising coal. The deslimed coking coal is separated into three products on a 3 m x 3.7 m Batac jig. The fines are froth floated in five 8 m^3 cells and concentrate is dewatered in two centrifuges. The steam coal is treated by a 7 m x 3.5 m Batac jig, from which the refuse is discarded and the coal product is crushed in an impact crusher and sized on vibrating screens. The 10 mm to 0.5 mm fraction is separated in six 600 mm dense medium cyclones. Sixty shaking tables treat 3 mm to 0.1 mm material and the minus 0.1 mm fines are froth floated in five 12m^3 cells. In this plant, all units, including the run-of-mine coal bunkers, are at ground level. The plant building is 220 m long x 65 m wide and 32 m high and of self supporting laminated wood rafter design.

Modern plants should be designed with energy efficiency in mind. Roberts showed that slurry transport may be made efficient by use of larger diameter pipes. Increase from 200 mm diameter to 250 mm gave 21% energy savings.[474] The greatest loss of energy in belt conveyance is caused by friction resulting from poorly aligned conveyor track, frayed belt edges, coal spillage and ungreased pulleys. Another source of energy saving is the use of more efficient, fluorescent tube lighting and photoelectric cells to switch off lights during bright daylight.

Excess noise should be avoided in plants since it interfers with communications between operators, prevents detection of other sounds which would warn of malfunctioning equipment and also is physiologically damaging to the operators. The U.S. Mining Enforcement and Safety Administration (MESA) has recommended that the daily noise exposure should not exceed 90 dBA constant level for eight hours duration 95 dBA for four hours and 100 dBA for two hours.[475] The dominant noise sources should be identified and modified as necessary. Noise reduction may be achieved by use of rubber or polyurethane screening surfaces instead of steel and lining feed chutes with rubber. When noise cannot be reduced at source, its propagation in the plant may be prevented by well placed sound absorbing screens.

2.26 Coal Preparation Practices

Zimmerman outlines current world trends in coal preparation.[476] Coal preparation practices vary very widely. Capacities of plants may range from 75t/h - 3000t/h. Plants may use very simple methods such as screening or they may be equipped with a very complex process flow sheet. They may be manually controlled or almost fully automated. Zimmerman distinguishes five levels of preparation:

1. crushing and screening only.

2. coarse coal cleaning only.

3. coarse coal and simple fine coal cleaning down to 0.06 mm.

4. coarse and fine coal cleaning and cleaning of the finer than 0.5 mm coal with closed water recovery circuit.

5. cleaning of all coal sizes in multiple stages of sizing and crushing for maximum liberation and optimum yield, producing two or more clean coal products, also including closed water recovery circuit.

It is estimated that by 1990 world production of coal will exceed 5000 Mt annually. Even if only 50% of the raw coal is cleaned, this represents an enormous tonnage to be treated. Consideration of coal preparation practices in other countries should be beneficial for Eastern Canadian coal by drawing attention to alternative methods which are proving successful elsewhere.

2.26.1 Canada

With the resurgence of the coal industry in the late 1960s and through the 1970s, coal preparation in Canada has become a complex and sophisticated as anywhere in the world.[477,478] While countries with longer technological histories in coal were able to evolve slowly, Canada was faced with the need for immediate innovative technology. Many of the earlier problems could be attributed to inadequate information on coal quality. Overoptimistic interpretation of data resulted in mine operators entering into contracts to deliver clean coal with unrealistically low ash contents. Plants had to lower their recoveries below the economic level to meet specifications. The friability of western Canadian coal was also seriously underestimated, resulting in inadequate fines circuits and undersize driers in most plants. These difficulties were only recognised and overcome with experience.

The main processes of coarse coal cleaning in Canada use dense medium baths (100 mm - 10 mm), dense medium cyclones (50 mm - 0.6 mm or 10 mm - 0.6 mm) and Baum jigs (50 mm - 0). Jigs cannot offer the same precision and efficiency as dense media separators but can process large tonnages at relatively low capital and operating costs. Clays present in the feed have less effect on jigs than on dense media.

A typical flow diagram for treating 37 mm to zero Canadian raw coal is given by Irwin.[479] The feed is first deslimed over screens and the screen oversize is treated in dense medium cyclones and dewatered in centrifuges. The slimes are separated in water-only cyclones, whose overlfow is classified in cyclones. The classifier cyclone underflow joins the clean coal, while the water-only cyclone underflow and classifier cyclone overflow are froth floated. A large proportion of the metallurgical coal, amounting to about 50% of the total production, is exported. The bulk of the thermal coal is for home use but exports to Japan are increasing.[480]

The characteristics of Canadian coals and their effects on coal preparation are outlined by Irwin et al.,[481] while Canadian practices are described by Butcher and Wright[482] and will not be summarized here.

The Quintette Coal Project, B.C., has reached start-up for the production of metallurgical coal for the Japanese steel industry and also some thermal coal. It is expected that by 1985 full production will be achieved, amounting to 5 Mt metallurgical coal and 1.38 Mt thermal coal annually.[483] Dense medium drum separation, dense medium cyclones, water-only cyclones and flotation are used in the treatment of metallurgical coal while the thermal coal is jigged and separated in water-only cyclones. Also in British Columbia, the Bullmoose Mine Project is starting-up to produce 2.3 Mt of metallurgical and thermal coal annually.[484]

2.26.2 United States

Currently about 40% of U.S. production is cleaned.[476] The use of jigging is still the most common method, accounting for 48%, while dense medium separation is used for 32%, shaking tabling 11.5% and froth flotation 5%.

Despite the great interest in fine coal cleaning world wide, Norton and Hambleton reason that from the producer's point of view, it does not pay to clean coal down to zero size, and there is a lower size limit to cleaning which provides maximum profitability.[485,486] Coal cleaning reduces yield and only in special circumstances can it achieve full heat content recovery. The best compromise, which satisfies both suppliers and users is to clean coal only to remove shale. The ash content of such a product would be that of the inherent ash of the coal seam. Since price structures are based on heat content, the removal of incombustible materials is economically advantageous to both parties. However, rejection of coal particles, even those with high ash will reduce the heat content of the clean coal with consequent loss in value. As an example, when a raw coal is cleaned to a yield of 81% with a heat content recovery of 95.6%, its price will increase from $US25/t to $US28.32/t, in proportion to the greater heat content/t. But the return on each tonne of raw coal produced falls to $US22.94, which is lower than the price of untreated coal. To obtain the same return for the clean coal as for raw coal, clean coal would need to sett for $US29.64/t. Thus, price structures based on a fixed rate per heat content do not compensate the producer for losses caused by cleaning, nor for the investment in coal preparation plant. Coal cleaning economics also depend on the moisture content of the product, since moisture has to be evaporated on burning, and thus, reduces the useful heat content. The moisture content of a 25 mm coarse coal is about 4$, while fine coal at 150 μm (100 mesh), may contain 25% moisture. Thus, to maximise its heat content, coal should be prepared at as coarse a size as possible. Crushing should be restricted to middlings intended for retreatment to minimise production of fines. The economically limiting size of coal cleaning varies with the nature of the raw coal. In the quoted example, the limit appears to be 150 μm. It is often better industrial practice to increase the separation density to accept higher ash content coarse coal, rather than to recover high moisture content fines. Accepting coarse middlings at 27% ash and 4% moisture is slightly more advantageous than recovering the minus 150 μm fines containing 14% ash and 32% moisture.

The coal preparation plant of Quarto Mining Company, built in 1972, employing simple jig washing and centrifugal dewatering at 1100 t/h, was extended to handle 1800t/h.[487] The finer than 6.3 mm coal, approximately 30% of raw feed, is now treated by cyclones and shaking tables. This scheme not only increased the capacity of the plant but also improved its overall efficiency since the fines fraction now bypasses the jigs.

An example of the treatment of high ash coal, (33%-40%), is illustrated by the 1200t/h coal preparation plant at Inland Mine No. 2, Sesser, Illinois.[488] The plant uses dense medium vessels, dense medium cyclones and froth flotation. Separating at 1.5g/cm^3 density gives 6.1% clean coal ash and reduces sulphur content to 0.9%.

2.26.3 United Kingdom

The present state and probable future of coal preparation in the U.K. are reviewed by Hillman.[489,490] About 95% of the coal in British mines is power loaded and the raw coal often contains 35% - 40% refuse and 80% is finer than

25 mm. Circuits were modified to treat coal with high refuse content. The fines cleaning sections of plants designed in the 1950s were often overloaded. Some of the fines were screened out and blended back with the clean coal for power station fuel, but this procedure often caused handling difficulties. Froth flotation is increasingly being used for the finer than 0.5 mm fraction. Today, over 6.5% is treated by froth flotation, about 60% is cleaned in jigs and 25% in dense media. Currently over 60% of production is used for power generation and about 15% as metallurgical coal. The reserves of high quality coking coals are very limited and increasing quantities of medium coking coals have to be produced by separating the floats at 1.3 - 1.4 g/cm^3. The sink fraction produces power station fuel after dense medium separation at 1.7 - 1.8 g/cm^3. Some developments in coal preparation in the U.K. are discussed by English,[491] [492] and Hillman.[490]

A paper by Knott of the National Coal Board, U.K. discusses coal preparation related to Canadian needs based on British expertise.[493] The paper outlines feed homogenization, choice of processes, fines treatment including solid-liquid separation, design aspects for safety and reliability and automation and computer control.

2.26.4 West Germany

The present state of coal preparation and possible future developments in the Ruhr area of West Germany have been outlined by Wilczynski.[494] The general trend is towards increased plant size, of around 1,000 t/h throughput, treating coal from more than one mine, and using large, more efficient machinery. The achievement of the greatest economy may be realised in the following ways:

1. The processing must aim at the highest possible clean coal yield. Since a yield increase of 1% gives income equivalent to between 25% and 35% of processing costs, an improved but slightly more costly form of treatment is easily paid for.

2. The processing plant capacity should be fully utilised to reduce the capital depreciation portion of the processing costs.

3. Sufficient coal should be stockpiled between the mine and plant and between plant and transport system to maintain smooth and trouble-free operations.

4. The preparation should be market orientated and the uniformity of products should be maintained.

Homogenisation of feed to the plant is also of utmost importance. Wilczynski compares the operation of conventional plants working with size fractions of 150 to 10 mm, 10 to 0.5 mm and minus 0.5 mm to hypothetical plants operating on 400 to 30 mm, 30 t0 2 mm, 2 to 0.1 mm and below 0.1 mm.[495] He states that the new treatment size limits would result in advantages over the established limits. The coarse tailings from the 400 to 30 mm fraction should be disposed of before homogenisation. The 30 to 2 mm fraction would be treated in dense medium with improved separation efficiency The 2 to 0.1 mm fraction should be treated on fine coal jigs and on shaking tables of multideck design. This size fraction would give a low sulphur product. The minus 0.1 mm fines could be froth floated at low solids concentrations in mechanical or pneumatic machines incorporating froth washing. Wilczynski compares the economics of the proposal and existing schemes and concludes that considerable increases in profits should result.

Leininger emphasises the necessity of homogenizing the feed since the resultant saving in capital cost of the preparation plant more than outweigh the cost of the homogenization facility.[496] Furthermore, homogenization gives improved selectivity of separation and increased clean coal yield.

Leonhardt surveys the current practices in coal preparation.[497] He discusses treatment schemes used in the U.S.S.R., U.S., U.K. and compares them with those employed in the West Germany. The paper includes the treatment of lignite, hard coal and coking coal, coarse and fine coal washing, cleaning coal with high soluble matter content, treating oxidised coals and the appropriate dewatering processes.

2.26.5 South Africa

Since South African coals are difficult to clean, of the treated coal only about 23% is by jigging, 55% by dense medium and 2.5% by flotation.[476] According to a survey, less than 2000 Mt of the coal reserves has ash contents between 10%-15%, about 10000 Mt between 15%-20%, 20,000 Mt between 20%-30% and 40000 Mt between 30%-35%.[498] Coals containing large proportions of inorganic constituents may be liberated, to some extent, by crushing. But the large proportion of near separation density material requires careful separation. With South African coals a remarkable reduction of ash content occurs in sizes below 100 μm. The necessity of crushing produces very fine coal fractions.[499] These fine fractions were discarded, but laboratory studies with oil agglomeration and froth flotation give acceptable results, although the high product moisture content needs further attention.

2.26.6 Australia

Australian coals tend to be geologically young and high in ash.[434] Their washability characteristics do not appear to be related either to their age or geographical location. In Australia 81% of bituminous coal was cleaned in1975.[500] About 45% is cleaned by jigging and the same amount by dense medium separation while the rest is treated by froth flotation and shaking tabling. Crushing is often used prior to separation to reduce the ash content of clean coal. On the whole Australian coals do not exhibit as much reduction in ash content at smaller sizes as do British and North American coals.[434] For thermal coals, adequate upgrading can be achieved with jigs on the coarser sizes and water only cyclones for the fines. For coking coals the separation frequently has to be made on a difficult part of the washability curve, requiring precision which can only be achieved with dense medium separation. In this case froth flotation is used for the fines. The proportion treated by froth flotation is increasing due to the trend towards reduced maximum size. This may simplify plant flowsheets by allowing the use of dense medium cyclones alone, dispensing with dense medium baths. Economically this may be undesirable as the minus 0.5 mm fines have to be froth floated at treatment costs of about two and a half times those of dense medium flotation. Australian coal preparation practices are discussed and illustrated with detailed statistics by Edwards.[500,501]

3. RECOMMENDATIONS

The recommendations of this report will be given in a coded form for simplicity.

Each topic is named and the relevent page number is listed. Other columns show current status of the technique, recommendations, comments and suggested priorities.

Notation

Current status - F - full scale.
P - pilot scale.
L - laboratory scale.
m - in mineral processing.
c - in coal preparation.

Recommendations - F - should be tested in full scale.
P - should be tested in pilot scale.
L - should be tested in laboratory scale.
R - research needed.

Comments - I - useful in intensive coal cleaning.
S - useful in pyritic sulphur reduction.
M - useful in coal-liquid mixtures.
D - useful in plant design or process control.

Suggested priorities - 1 - considerable importance.
2 - moderate importance.
3 - negligible importance.

Subject	Page	Status	Recommendations	Comments	Priority
Liberation studies	3 - 7	Lm,Lc	L	I,S	1
Revolving grate impact crusher	10 - 11	Pm,Pc	P		2
Roller screen crusher	10 - 11	Fc	F		2
Ring and ball mill	14	Fm,Fc	F	I,S	2
Cage mill	14	Fm,Fc	P	I,S	2
Szegö mill	14	Pm,Pc	P	I,S	2
Autogenous mill	16	Fm	P	I,S	3
Vibrating mill	16 - 18	Fm	P	I,S.M	2
Tower mill, Paddle mill	18 - 19	Fm,Pc	P	M,S	1
Ore-bed magnetic lining	20 - 21	Fm	F		2
Grinding aids	22	Fm	F,L,R	M,	1
Multi-angle deck screens	24	Fm,Fc	F		2
Mogensen sizer	24	Fm,Fc	F		2
Rotating probability screen	25 - 27	Fe	F		2
Air-jet screening	28	Pm	P,R		2
Bauer-Hydrasieve	32 - 33	Fm,Fc	P		2
Wallis ultrasonic separator	33 - 34	Fm	P		3
Mathematical model for screening	35 - 36	Pm	P	D	2
Hydrocyclone simulation	43	Fm	P	D	2
Small diameter hydrocyclones	44	Fm	P		2
Choking-free cyclone	44 - 45	Fm	P		3
Hydrocyclone filter	45 - 46	Fm	F		2
Two stage classification	47 - 48	Fm	P	M	1
Individual mineral behaviour in classification	48 - 49	Lm	L,R	I,S	2
Centrifugal deduster	49 - 51	Fc	P		2
Batac jig	56 - 57	Fc	F		3
Fine coal jig	57 - 58	Fc	P		2
Shaking table	61 - 65	Fm,Fc	P	I,S	2
Reichert spiral	65 - 68	Fm,Fc	P		1

Subject	Page	Status	Recommendations	Comments	Priority
Reichert cone	68 - 71	Fm	P	I,S	1
Counter-current fluidized cascade	73 - 75	Pc	P	S	1
Dense medium viscosity control	79 - 80	Pm	L,P,R		1
Density tracers	81 - 82	Fm	F	D	1
Effect of particle size in centrifugal dense medium separation	83	Lm	L,R		2
Swirl cyclone	86 - 87	Fc	P		2
Three product cyclone	87	Fc	P		3
Vorsyl separator	88 - 89	Fc	P		3
Dynawhirlpool separator	89 - 90	Fc	P		3
Tri-Flo separator	91 - 92	Fm	P		1
Stripa process	92 - 95	Fm	P		2
Otisca process	95 - 97	Pc	P	S	2
Liller cyclone	90 - 100	Pc	P	S	1
Samaruim-cobalt magnets	103 - 104	Fm	P		1
Magnetic hydrocyclone thickener	104 - 105	Pm	P		2
High gradient magnetic separation	105 - 106	Fm,Pc	P	S,M	2
Superconductive magnetic separation	107	Lm	L		3
Magnetohydrostatic separation	107 - 108	Lm	L	S,M	3
Magnex process	108 - 109	Pc	P		2
Separation with magnetic fluids	109	Pc	P,R		2
Conductivity separation	110 - 112	Fm,Pc	P		3
Contact potential separation	113 - 114	Lm,Lc	L	S	3
Dielectric separation	114 - 116	Lm	L	S	3
Sorting	116 - 126	Fm	R		1
Oxylore process	121 - 122	Pc	P		2
Effect of particle size in flotation	132 - 137	Lm,Lc	R		1
Effect of oxidation in flotation	137 - 128	Lm,Lc	R		1
Effect of slime coating	138 - 140	Lm,Lc	R		1
Electrokinetic potential studies	140 - 141	Lm,Lc	R		1
Intermittent aeration	142 - 143	Pm	P		2
Split conditioning	143 - 144	Fm,Fc	P	S	1

Subject	Page	Status	Recommendations	Comments	Priority
Large flotation machines	146 - 148	Fm,Fc	F		1
Cyclo-cells	149	Fc	F		2
Davcra cell	149 - 150	Fm	F		2
Klausthal cell	150 - 151	Pc	P		2
Flotation column	151 - 152	Fm	P		2
Leeds flotation column	152 - 154	Fc	P		2
Reciprocating plate flotation column	154 - 155	Lc	L,R		1
Froth separation	155 - 156	Fm	P		2
Static flotation machine	157 - 158	Fm	P		2
Air sparged hydrocyclone	159 - 160	Pc	P		2
Electroflotation	160 - 161	Pm	L		3
Leeds open-top laboratory cell	161 - 162	Lc	L		1
Modified Denver Model D1 cell	162 - 163	Lc	L		1
Modified Denver Model 12 cell	163	Lm	L		2
Flotation models	163 - 164	Pm,Pc	F	D	1
Flotation circuit analysis	165 - 166	Fm,Fc	F	D	1
Selective flocculation	166 - 168	Fm,Lc	L	S	2
Oil agglomeration	168 - 171	Pc	P	S,M	1
Coal-liquid mixtures	171 - 172	Pc	P	S,M	1
Pyritic sulphur removal	172 - 174	Fc	F	S	1
Separation by coal petrography	174	Lc	L,R		2
Flocculation in solid-liquid separation	174 - 180	Fm,Fc	F,L,R		1
Thickening test	180 - 181	Lm,Lc	F,P,F,R		1
High capacity thickeners	181 - 183	Fm,Fc	F		2
Lamella thickeners and modules	183 - 184	Fm,Fc	F		2
Deep cone thickeners	184 - 185	Fc	F		2
Screen bowl centrifuges	187 - 189	Fc	F		2
Dehydrum and dehydrator	189 - 190	Pc	P		2
Filtration testing	190	Lm,Lc	L,R		1
Double drum rotary filter	191 - 192	Fc	F		3
Elimination of filtrate pumps	192	Fc	F		2

Subject	Page	Status	Recommendations	Comments	Priority
Oil assisted filtration	192	Pc	P,R		2
Steam assisted vacuum filtration	192 - 193	Fc	F		2
Electrofilter	193	Fm	P		3
Belt filter press	193 - 194	Pc	P		1
Membrane filter plates	194	Fc	P		2
Automatic pressure filters	194 - 197	Fm	P		1
Filter cake dewatering aids	197	Fm	F,P.L,K		1
Tailings stabilization with cement	199	Fc	P		3
Reclamation of coal from waste	200	Fc	P		2
Computerized coal washability determination	202			D	2
Use of process nodelling	204 - 207		F	D	1
Improvements in process control	207 - 209	Fm,Fc	F,R		1
Ash and sulphur monitoring	209 - 210	Fc	F,R		1
Particle size monitoring	210	Fm	F	I,S,M	2
Vibration and noise analysis	210 - 211		F		1
Modular plants	212 - 213	Fc	F		2
Low profile plant buildings	213 - 215	Fc	F		2
Energy saving methods	215	Fc	F		1
Noise reduction	215 - 216	Fc	F		1
Evaluation of coal preparation practices used abroad	216 -				1

These recommendations have been made after consideration of the technical and, where available, the economic improvements, that the techniques or equipment would offer. Those items which affect a greater part of the plant operation are considered as the more important. Implementation of some of the recommendations should result in considerable benefits to coal preparation. Since the relative merits of various items are the author's personal opinion there is need for consideration of the recommendations by other persons.

4. ACKNOWLEDGEMENTS

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